TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

(INCORPORATED)

1931

GENERAL VOLUME

THIS VOLUME CONTAINS PAPERS AND DISCUSSIONS PRESENTED AT MEETINGS HELD AT SAN FRANCISCO, OCTOBER, 1929; EL PASO, OCTOBER, 1930; NEW YORK, FEBRUARY, 1930 AND 1931; AND BOSTON, SEPTEMBER, 1931; OFFICERS, COMMITTEES, NECROLOGY, ABSTRACTS, TITLES OF PAPERS IN 1931 VOLUMES, LIST OF 1931 TECHNICAL PUBLICATIONS AND CONSOLIDATED INDEX

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PREFACE

This volume of TRANSACTIONS is the fifth and final to be issued by the Institute during 1931. The preceding volumes were:

TRANSACTIONS, Petroleum Development and Technology, 1931.
TRANSACTIONS, Institute of Metals Division, 1931.
TRANSACTIONS, Coal Division, 1931.
TRANSACTIONS, Iron and Steel Division, 1931.

With respect to papers of a technical character, the present volume includes those available on the following subjects:

Metal Mining	Mining Geology
Nonferrous Metallurgy	Gold Supply Symposium
Nonmetallic Minerals	

The only residual TRANSACTIONS material is the group of papers on Geophysical Prospecting that will appear in a special volume early in 1932, and several papers coming under the head of Milling and Concentration that are being held in anticipation of a future special volume.

The present volume serves two other purposes:

(*a*) In it, as a matter of permanent record, are published the roster of officers and principal standing committees for the year; brief summaries of the proceedings of the Annual Meeting in New York and of the Division and Regional Meetings held during 1931; and the necrology for 1930.

(*b*) It constitutes a complete index of the publications of the Institute for the year. In addition to a consolidated index of TRANSACTIONS, TECHNICAL PUBLICATIONS, and MINING AND METALLURGY, the monthly magazine, are to be found a classified series of abstracts of new papers by means of which the scope and character of any contribution may be determined. In addition, are condensed contents pages of each volume of TRANSACTIONS and a classified list of TECHNICAL PUBLICATIONS.

Each member is provided with this general volume, in addition to which he is entitled to one of the volumes mentioned above, in accordance with the selection he has already made. Any of the others is supplied on request at the cost price of $2.50. All volumes are now furnished in cloth binding, without extra cost.

As members of the Institute, our thanks are due first to those who have given their time, energy and ability to preparing the papers that comprise this set of five splendid volumes; second, to the members of the Committee on Papers and Publications, who are among the most earnest workers in the organization; and third, to H. Foster Bain, who save for his retirement on November 1, would be writing this preface as Secretary of the Institute.

A. B. PARSONS, *Secretary.*

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* Mr. Bain resigned as Secretary, in the fall of 1931, and Mr. Parsons was appointed by the Directors to that office. The change went into effect on November 1.

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* One representative each from the Blast Furnace & Coke Association of the Chicago District and of the Eastern States Blast Furnace & Coke Oven Association cooperates with this committee.

New York Meeting

The 140th meeting* of the American Institute of Mining and Metallurgical Engineers was held in New York, Feb. 16 to 19, 1931. It consisted of the annual business session, twenty-six technical sessions at which 149 papers were presented, two round table discussions, two meetings of the Board of Directors, three meetings of Section Delegates, two formal lectures, four group dinners and 11 group luncheons, luncheon daily for members and guests, 20 committee meetings and conferences, the annual reception and dinner dance, an informal dance, a smoker-dinner, two meetings of the Woman's Auxiliary and a special program of entertainment for the ladies. Former members of the Rocky Mountain Club and friends lunched together at the Engineers Club on Tuesday.

The proceedings of the Iron and Steel Division included four general technical sessions, an Iron Ore Round Table, the Howe Memorial Lecture, the Division luncheon at the Engineers Club Wednesday noon, a session each of the Executive Committee and the Committee on Nondestructive Testing, a luncheon each of the Committee on the Manufacture of Alloy Steel and the Blast-furnace Conference Group, and a dinner of the Committee on Physical Chemistry of Steelmaking. Dr. F. F. Lucas delivered the Howe Lecture, the title of which was "On the Art of Metallography."

The Institute of Metals Division held three general sessions, its annual lecture, which was delivered by Dr. Arne Westgren and entitled "X-ray Determination of Alloy Equilibrium Diagrams," and a Division dinner at the Hotel New Yorker Thursday evening, at which Dr. G. W. Thompson was the principal speaker. The Executive Committee assembled at a luncheon on Thursday.

A round table on Coal Land Valuations, a general technical session and a Division luncheon were held by the Coal Division. C. E. Bockus, President of the National Coal Association, made an informal talk at the luncheon which was held at the Engineers Club Monday noon.

The Petroleum Division held six general sessions, including its annual production symposium. The annual dinner was held at the Engineers Club Thursday evening.

A Gold Supply Symposium was the feature of the proceedings of the Committee on Mining Geology. This Committee also held a general session and a luncheon meeting. Sessions were held by other Technical Committees as follows: Mining Methods, 2; Mine Ventilation, 1 (besides which the Committee on Mine Ventilation Code held a meeting); Ground Movement and Subsidence, 1; Milling Methods, 1; Nonferrous Metallurgy, 1; Engineering Education, 1 (also a Committee luncheon); Nonmetallic Minerals, 1, and Geophysical Prospecting, 2.

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* For preliminary story of meeting see MINING AND METALLURGY (Feb., 1931) 81; for news story see the March, 1931, number.

At the annual business meeting on February 17, reports of the President, Treasurer and Secretary were presented, and the following ticket was elected: Robert E. Tally, President; Howard N. Eavenson, Vice-president; H. A. Guess, Vice-president; L. S. Cates, Karl Eilers, S. R. Elliott, H. G. Moulton and William Wraith, Directors. At an executive session of the Directors on Tuesday afternoon, Karl Eilers was elected to succeed himself as Treasurer, and H. Foster Bain to succeed himself as Secretary.

Twenty-two Sections and the four Divisions sent delegates to the Annual Meeting. These delegates held three sessions and were guests at the Directors' dinner on Tuesday evening.

The annual meeting of the Woman's Auxiliary was held on Tuesday. A theater party, a television demonstration and a recital were features of the program of entertainment during the week.

The regular Tuesday evening informal dance was held this year at the Grand Central Galleries. It was the best attended in the history of this event.

A successful novelty at this meeting was the dinner-smoker. It was held at the Hotel Pennsylvania on Monday evening with more than 600 in attendance. A five-course dinner preceded the smoker and an excellent program of entertainment was the feature of the latter.

The annual dinner dance, held Wednesday evening at the Hotel Commodore, was well attended. The incoming and retiring presidents held a reception preceding the dinner. Ralph M. Roosevelt, chairman of the New York Section, was toastmaster. Honorary Membership in the Institute was bestowed upon Dr. Waldemar Lindgren. The members of the Class of 1881 of the Institute Legion of Honor were introduced, Ralph Crooker responding for the group. The James Douglas medal was presented to William H. Peirce. The William Lawrence Saunders medal was presented to F. W. Maclennan. E. S. Davenport was the recipient of the Robert W. Hunt prize. Retiring President William H. Bassett presented his presidential address.

Fairmont Meeting

The Coal Division met* at Fairmont, W. Va., March 26 and 27, with approximately 125 registered. The Fairmont Hotel was headquarters. There were two sessions on the first day, at which were presented seven papers. In the evening a dinner was held at the hotel, at which the speakers were N. A. Elmslie, Mayor H. G. Martin of Fairmont, President John R. Turner of West Virginia University, R. M. Lambie, Harrington Emerson and others. The second day was spent in a

* For news stories of meeting see MINING AND METALLURGY (April, 1931) 175 and (May, 1931) 237.

morning inspection of the Carolina mine of the Consolidation Coal Co. and after lunch in a visit to the plant of the Domestic Coke Corporation.

Directors Meeting at Wilkes-Barre

The Board of Directors held its regular monthly meeting on May 22,* at Wilkes-Barre, Pa., preceding a dinner given there by the Pennsylvania Anthracite Section on the occasion of the Sixtieth Anniversary celebration of the Institute.

Boston Meeting

The Institute of Metals and Iron and Steel Divisions met† jointly in conjunction with the American Society for Steel Treating, at the Statler Hotel, Boston, during the week of the National Metal Congress, September 22 to 26. The Institute registration was 324 members and guests. The Institute of Metals Division held two sessions, one on Copper and Copper Alloys, the second a general session. The Iron and Steel Division held a general session and a session on Alloys of Iron. The two Divisions held three joint sessions: a session on Age-hardening, the Science Lecture, delivered by Dr. P. W. Bridgman, and a symposium on Metallurgical Education. The Iron and Steel Division and the American Society for Steel Treating held a joint session on Nitriding and Carburizing. There were 26 technical papers presented by the two Divisions. Dr. Bridgman's lecture was entitled, "Recently Discovered Complexities in the Properties of Simple Substances."

The joint dinner of the two Divisions was held at the Hotel Statler on Wednesday evening. Alan Kissock gave a nontechnical talk on "Molybdenum." The Executive Committee of the Institute of Metals Division held a luncheon meeting Tuesday noon and the Executive Committee of the Iron and Steel Division a luncheon meeting at noon on Wednesday.

The Papers and Publications Committee of the Iron and Steel Division held a luncheon meeting on Thursday, at which it selected the contents of TRANSACTIONS, Iron and Steel Division, 1931.

An unusual number of trips was offered: Tuesday morning, an inspection trip to Watertown Arsenal, Watertown, Mass.; Wednesday morning, inspection trip to Bethlehem Shipbuilding Corpn., Fore River Plant, Quincy, Mass., or a combined trip to the Boston Gear Works, Norfolk Downs, Mass., and Pneumatic Tool Co., Norfolk Downs, Mass.; Thursday morning, inspection trip to General Electric Co., West Lynn, Mass.; or to Naumkeag Steam Cotton Co., Salem

* For news story of meeting see MINING AND METALLURGY (June, 1931) 292.
† For news story of meeting see MINING AND METALLURGY (Nov., 1931) 480.

Mass.; Friday morning, inspection trip to United Shoe Machinery Co., Beverly, Mass., or to Trimont Mfg. Co., Roxbury, Mass.

Joplin Meeting

The Institute joined with the Western Division of the American Mining Congress in holding a joint meeting* at Joplin, Mo., on September 28, 29 and 30. The Hotel Connor was headquarters. The Institute Committee on Engineering Education held two sessions, Sunday afternoon and evening, September 27. Monday was A. I. M. E. Day and five technical papers were presented: by J. C. Heilman,[1] C. K. Leith,[2] H. A. Buehler,[3] W. M. Hayward and W. H. Triplett,[4] and George M. Fowler and Joseph P. Lyden,[5] respectively. Monday evening was given over to entertainment in the roof garden of the Connor Hotel.

Tuesday was Mining Congress Day and President Robert E. Tally was one of the speakers at the afternoon session; his subject, "The Mining Outlook." At the banquet on Tuesday evening Charles A. Neal was toastmaster. Missouri State Senator Cliff Titus was the principal speaker. Secretary H. Foster Bain spoke briefly on "Experiences in China."

Wednesday was Excursion Day and the entire day was spent in the field. The first stop was made at the Blue Mound, from which the whole of the Picher field was surveyed. An underground trip was made through the Blue Goose and Angora mines of the Commerce Mining & Royalty Co. and the Shorthorn mine of the Bilharz Mining Co. The route led through Galena, Baxter Springs and Picher to Miami, where a barbecue lunch was provided. Following lunch, boys of the district, organized as Mounted Scouts of America, gave a junior rodeo. Returning, a stop was made to inspect the diesel power plant of the Commerce company at Cardin, Okla., and to look over the newly completed Bird Dog mill of the same company.

Entertainment for the ladies included a trip through the Ozarks on Monday and on Wednesday they were taken over the same route as the inspection trip for the men.

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* For news story on meeting see MINING AND METALLURGY (Nov., 1931) 476.

[1] J. C. Heilman: Mining and Economic Conditions in the Tri-State District. *Min. & Met.* (Sept., 1931) 392.

[2] C. K. Leith: Problems of Mineral Surplus. *Min. & Met.* (Nov., 1931) 472.

[3] A talk on "Geology and Ore Deposits of the Ozark Region" which is to be prepared in technical paper form from the stenographer's notes.

[4] W. M. Hayward and W. H. Triplett: Occurrence of Lead-zinc Ores in Dolomitic Limestones in Northern Mexico. A. I. M. E. *Tech. Pub.* 442.

[5] G. M. Fowler and J. P. Lyden: The Ore Deposits of the Tri-State (Missouri-Kansas-Oklahoma) District. In preparation as A. I. M. E. *Tech. Pub.* 446.

Year of Election	Name	Date of Death	Issue Containing Biography 1930	Pag
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* 1931 Issue.

† A Rocky Mountain Club member of the Institute.

Recently Discovered Complexities in the Properties of Simple Substances*

By P. W. Bridgman,† Cambridge, Mass.

(Boston Meeting, September, 1931)

It is a commonplace that experimental physics in the last few decades has discovered manifold complexities in the atomic and subatomic levels, where it was thought for hundreds of years that no structure existed, as is witnessed by the derivation of the word "atom" itself. The field of exploration opened by the discovery of the electrical structure of the atom was so fundamental, rich and stimulating, that it has been almost exclusively cultivated by physicists since they obtained their first intimations as to the nature of the basic facts. Only within the last few years has it begun to dawn on us that we have overshot an enormous domain in which are situated many phenomena of fundamental importance for all the practical uses to which we put matter in our daily lives; the domain, that is, of most of the phenomena of interest to the biologist and the metallurgist. Furthermore, we are beginning to find that this intermediate domain is not entirely simple, but that it contains unsuspected complexities, some of them derived from the complexities of structure on the atomic and subatomic levels, and some of them emergent as matter collects itself into large aggregates, which often offer the key to the explanation of hitherto baffling large-scale properties of matter. Today I want to describe and consider some of these newly discovered complexities in the behavior of matter in bulk. The application to metallurgical problems is not always direct, but I hope to be able to suggest that such general notions as to structure as those here discussed must have important reactions on our attitude toward the problems of metallurgy.

Three different sorts of phenomena of this kind to which I shall direct your attention are (1) the so-called structure sensitive phenomena, (2) complexities of such a nature that they are masked by the presence of extraordinarily small amounts of impurities, and (3) complexities depending on internal molecular rearrangements which can be understood only from the point of view of the quantum theory.

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* Science Lecture delivered before the Institute of Metals Division and the Iron and Steel Division of the A. I. M. E.

† Hollis Professor of Mathematics and Natural Philosophy, Harvard University.

Structure Sensitive Phenomena

First let us consider the structure sensitive phenomena. It has been known for a long time that some properties of matter are much more definite and clean-cut than others, in the sense that they are easy to measure, and measurements made on different specimens by different observers under different conditions and by different methods yield consistent results, so that such results can justly be regarded as characteristic of the material under investigation. Perhaps the most striking of such properties is the atomic weight of some of the elements; ordinary mechanical density, lattice structure, heat of solution, and optical dispersion are other examples. There are other properties which are less definite, and numerical agreement by different observers is much more difficult to obtain. Sometimes there are technical difficulties in devising the apparatus or in making the measurements; as, for example, thermal conductivity measurements, which are notoriously difficult because of the impossibility of obtaining thermal insulators by which the flow of heat can be compelled to take only the desired channels. Other such effects are the various transverse effects in a magnetic field, such as the Ettingshausen effect, which are exceedingly difficult to measure because of their extreme smallness, and because of the difficulty of eliminating disturbing secondary effects. Sometimes, with the improvement of technique, properties which were at first difficult to measure and which gave conflicting results have become easy and consistent. An example of this sort of thing is cubic compressibility, for which the most discordant results are recorded in the early literature. The difficulty was both in the theory, which at first was not at all well understood, and also in the measurements themselves, because the effects are very small. But the technique has been so improved that now different observers with different apparatus and material from different sources can obtain consistent results, and we can now be assured that recent experimental values for cubic compressibility are truly characteristic of the material.

But apart from difficulties arising from difficulties of technique, there are outstanding certain physical properties for which it is very difficult to get consistent results on material from different sources. Thus although different observers do not find it difficult to agree on the proper value for Young's modulus of iron, various results are to be found for the elastic limit and the ultimate strength. There are many phenomena in the same category, for example, electrical conductivity, diffusion and mixing or unmixing phenomena in the solid state, such optical properties as photoelectric absorption and the electrical conductivity induced in crystals like rock salt by photoelectric action, magnetic susceptibility, and the effect of tension on thermoelectric

quality. These properties may be greatly affected by variations in the heat treatment, conditions of crystallization, mechanical working, or by slight impurities.

The first clear recognition that physical phenomena fall into two such groups of sensitive and insensitive properties, and that the sensitive properties demand for their explanation new sorts of consideration not entertained in the ordinary physical theories, was doubtless due to Smekal. The effects are particularly striking with regard to the ultimate strength of single crystals, for example, of a material like NaCl, to take an example not greatly complicated by plastic flow. Certain of the properties of NaCl, as, for example, its heat of solution, or its compressibility, can be calculated theoretically from the lattice structure of the crystal as revealed by X-rays. The calculation involves the assumption of centers of force at the nuclear points indicated by X-rays, and the fact that the results of the calculations agree fairly well with the experiments is pretty good evidence that we have approximately a correct understanding of the situation so far as these phenomena are concerned. But if we know the forces between the ions with sufficient accuracy to calculate the lattice energy and the compressibility, we should also be able to calculate the breaking strength, because this involves only a complete knowledge of the interionic forces when the ions are separated by a great distance. These calculations were made and a result found of the order of 1000 times greater than the experimental values. It was for a time thought that the low experimental values could be explained by the effect of surface imperfections, but after a great deal of discussion it now seems to be accepted by most investigators that the way out is not to be found here, but that the strength of NaCl is enormously lower than it ought to be from considerations which take account merely of its lattice structure.

Smekal recognized that some new physical consideration is demanded to account for the enormous discrepancy between this theory and experiment, and he proposed the new theory that in an actual crystal the lattice structure does not extend indefinitely in every direction, but that it is broken by faults or imperfections into blocks of anywhere from 10^4 to 10^6 atoms each, and that in the places where the blocks fail to join perfectly there are cavities of atomic dimensions left in the crystal structure. The faults between the blocks and the cavities are thought to be responsible for the great discrepancy between some sorts of calculated and experimental phenomena. It is easy to see why certain properties should not be affected by these faults and why the experimental values of these properties should agree with the simple theory. The density, for example, is determined by the mean distance of separation of the atomic units, and since the faults affect only a few of the atoms, the mean distance, and so the density, can be only little affected.

Similarly it is evident that the energy of the lattice, as given, for example, experimentally by the heat of solution, will be little affected by the faults, since the relative position of most of the atoms is unaffected, which means that the simple lattice theory will continue to give the essential explanation of the situation, even when the faults are considered. It is, on the other hand, evident that a phenomenon like the tensile strength of a brittle crystal, to take an example in which complications are not introduced by plastic flow, should be profoundly affected by the internal faults. The mathematical theory of elasticity shows that in the neighborhood of a reentrant angle, or in an interior cavity with sharp angles in a solid material which is the seat of elastic stresses, the stress may build up to infinite values, merely because of geometrical considerations, although the average stress through the body of the solid is finite. In practice this means that a reentrant angle is a source of great weakness, and that rupture takes place in such solids very much more easily than in a geometrically sound piece of the same material. Rupture, in such a material with internal faults, is propagated by a crack, the advancing head of the crack always remaining sharp, and thus providing for the continual existence of the necessary reentrant angle. Theoretically, a perfectly homogeneous piece of a material carrying an imperfection of this sort would rupture with a vanishingly low stress. Actually, substances are not perfectly homogeneous and never have mathematically sharp angles because of the limitations imposed by atomic structure, if for no other reason, but in any event it is easy to see that the existence of imperfections of this general nature will greatly lower the stress needed to produce rupture, and thus account for the discrepancy between the theoretical and the experimental values of the breaking strength.

Relation of Faults to Sensitive Properties of Matter

Smekal has considered a large number of the "sensitive" properties of matter, and has shown in detail how the existence of the faults explains their behavior. His considerations have been mostly confined to nonmetallic crystals, of which rock salt is typical. For example, Gudden and Pohl found that in rock salt there are two kinds of optical absorption present simultaneously; one kind of absorption, by far the most important, is a long-wave absorption, and is to be accounted for by the excitation of the atoms in the conventional way. This part of the absorption is to be thought of as contributed by the unfaulted blocks. Superimposed on this there is a much weaker absorption toward the short-wave side of the spectrum, corresponding to the ordinary photoelectric effect, and which is due to the incident light pulling electrons out of the chlorine ions which abut on the sides of the cavities. These electrons, which are detached in this way inside the cavities, immediately recombine

with the positively charged sodium ions which also abut on the cavity walls, and in this way produce neutral sodium atoms. The presence of such neutral atoms imparts to the crystal of NaCl the characteristic purple "radiation" color, so that one has here a most beautiful way of detecting the existence of these cavities and studying the way in which they are affected by various external factors. One would expect, for example, the number of the cavities to be greatly affected when the crystal is strained beyond its elastic limit. If one bends a bar of NaCl enough to give it plastic deformation, it follows from elasticity theory that there will be a relatively unstrained region in the center, and that the upper and lower faces will be permanently affected, the one being plastically deformed in extension, and the other in compression. Such a deformed bar, when illuminated by X-rays to bring out the color, does indeed show a marked coloring of the two deformed faces, while the relatively unstrained center receives only a relatively light coloring. Or if the bar is first colored by being radiated and then deformed, it will be found that the color disappears in the plastically deformed faces, but remains in the elastically unharmed central portion. The explanation is that the deformation produces relative motion in the walls of the cavities, so that the neutral atoms on the walls of the cavities have an opportunity to take their places again as ions in the regular lattice structure, thereby losing their color.

One would expect that the production of electrons in the cavities would have an effect on the electrical conductivity, and this turns out to be the case. Smekal has shown how to analyze the electrical conductivity of NaCl (which is always, of course, exceedingly small) into two parts, one of which is constant irrespective of the degree of purity of the crystal and its thermal treatment, and is to be considered characteristic of the pure lattice structure, while the second part varies greatly from specimen to specimen and is to be ascribed to the free electrons associated with the cavities.

The cavities or imperfections should change in number and importance as the material changes. In the first place, they should be intimately connected with impurities, because every atom of impurity in a crystal usually constitutes a place where the perfect regularity of the crystal lattice is disturbed, and therefore is a potential location for a cavity. It does indeed seem to be true in general that the sensitive properties change in the way that would be expected when the amount of impurity is decreased. However, there are difficulties here because of the difficulty in any practical case of reducing the amount of impurity sufficiently to separate the effect due to it from other effects. It was suggested above that the blocks in the secondary structure might contain possibly 10^6 atoms. Now it is practically impossible in most cases to reduce the impurity to less than one part in 10^6, so that perhaps we have

not yet been able to really separate the effect of the other factors from the effect of impurities. There is room for much important future work here, and we are just beginning to recognize that the effect of impurities, which by the old standards would have been considered absolutely negligible in amount, may be very important. There seems to be no doubt that the impurities have a tendency to separate into the faults between the blocks, and in this way to exert a disproportionately great effect on the sensitive properties.

There is also a close connection between the size of the blocks and the heat treatment. It is at first sight somewhat paradoxical that the secondary structure has a more important effect in an NaCl crystal grown by slow solidification from the melt than in one grown from aqueous solution. One would expect microscopic inclusions of the mother liquid to remain in the crystal grown from solution, and so the effect of imperfections to be larger. The fact seems to be that the size of the secondary structure is in some way connected with the thermal fluctuations of density to which the crystal was subject during the process of growth, which are satisfactorily explained by ordinary statistical theory. These statistical fluctuations are greater at higher temperatures, and therefore will give rise to greater effects in the secondary structure in crystals produced from the melt at high temperatures than in those produced from cold aqueous solutions. This point of view may be checked by annealing at high temperature a crystal grown from aqueous solution, when it will be found that the crystal acquires the secondary properties of a crystal deposited from the melt.

Secondary Structure

The assumption of secondary structure makes understandable many hitherto puzzling phenomena. Consider, for example, the complicated phenomena of elastic after-effects and elastic hysteresis. Artificial theories of these have been given in which the material has been supposed to have a complex structure, being composed of different sorts of elementary materials with different properties combined in complicated ways. Thus theories have been made of the elastic after-effects in which mechanisms have been imagined equivalent to perfectly elastic springs tethered to material with the viscosity of molasses. Evidently a much more satisfactory basis is provided for such phenomena by finding the complexities in the small-scale geometrical structure rather than in the properties of the material itself. Thus there would seem to be plenty of opportunity in the neighborhood of the faults of the crystal for those internal strains and local plastic effects necessary to hysteresis and elastic after-effects.

Other sorts of phenomena are also consistent with this point of view. For instance, the fact that thermal conductivity is a sensitive property

falls in at once with the modern picture of thermal conductivity as due to elastic waves of microscopic dimensions. We would expect these waves to be scattered by the faults, and so the thermal conductivity to be affected. The picture is also consistent with the fact that electrical conductivity is a sensitive property. The picture that we now have of electrical conductivity involves the assumption that the electrons have what would have been described from the old classical point of view as long free paths. *Long* free paths can be affected by comparatively infrequent faults in the crystal, whereas if the paths are short such faults will be of comparatively little importance; in this way we can see how it is that electrical conductivity can be sensitive.

As already stated, the work of Smekal on this subject was confined mostly to nonmetallic crystals, like NaCl; here the experimental verification is comparatively easy because of the existence of such phenomena as coloring by X-rays. The extension to metallic substances is much more difficult. Very important work on this aspect of the subject is now being done at Pasadena, at the California Institute of Technology, where there is activity on both the experimental and theoretical sides.

On the experimental side, Goetz has made painstaking and elaborate studies of the properties of crystals of bismuth, a material in which the secondary effects are particularly large, and which is therefore a favorable material for investigation. It has been known for a long time that many of the properties of bismuth are exceedingly variable from specimen to specimen, and therefore are presumably structure sensitive. The electrical conductivity is such a property; it is sensitive to small amounts of impurity and is also sensitive to variations in the direction of current flow in the crystal. The conductivity perpendicular to the crystal axis, that is, for directions of current flow in the basal plane, which is the plane of principal cleavage, proves not to be sensitive, and different observers agree rather well on this value; whereas if the current flow is parallel to the crystal axis, which means across the cleavage planes, the disagreement between competent observers may rise to as much as 30 per cent, the conductivity in this direction being less the greater the impurity or the greater the chance for incipient fractures on the cleavage planes. The temperature coefficient of conductivity is also extremely sensitive, a few hundredths of a per cent of lead being sufficient to reduce it to one-half its normal value. Another similar phenomenon is the extraordinary sensitiveness of bismuth crystals to minute stresses during the process of crystallization already studied by Kapitza.

Goetz has used mostly two methods for controlling the secondary structure. In the first place he has altered the conditions during growth by crystallizing from the melt with or without the presence of a magnetic field. The primary properties of the crystal, such for example, as its lattice structure as shown by X-rays, are entirely unaffected by the

presence of the magnetic field, and are characteristic only of the material. But other properties, of which the thermoelectric behavior has been specially studied, depend greatly on the field which was applied during crystallization, and therefore are to be classed as structure sensitive. In the second place, the purity has been altered, and some of the properties, as already known, found to vary tremendously with the presence of slight amounts of impurity. The effect of combining an impurity with a magnetic field during crystallization is most complicated. Doubtless the most interesting and important contribution of Goetz is to prove the actual physical existence of the secondary structure on which the sensitive properties are supposed to depend. The structure consists of networks of planes so cutting the basal planes as to give a pattern of equilateral triangles. This substructure can be brought out by suitable etching, and can be shown by photomicrographs, or it can be studied in a most convincing way by watching under a microscope the process of solution in an electrolyte, when it will be found that periodic slight changes are necessary in the solution voltage, corresponding to the arrival of the surface of solution at each of the secondary planes. The most curious feature of this secondary structure is that the spacing of the planes appears to be definite, independent of the conditions of growth or the degree of impurity. There seems to be no doubt that the impurities segregate themselves on the secondary planes; if the amount of impurity is increased, the number of atoms of impurity which separate on any plane increases, accordingly, but the number of planes does not change.

Diamagnetic Susceptibility

Another interesting fact found by Goetz is that the diamagnetic susceptibility is also structure sensitive in a way similar to the electrical conductivity, only here the susceptibility for the magnetic field parallel to the axis is not sensitive, while for the perpendicular direction of the field there is sensitiveness. This again is consistent with recent views as to the nature of diamagnetic action, which is thought not to be an atomic affair, but to involve the cooperation of many atoms, the magnetic electrons effectively describing orbits many atoms in diameter. If these orbits cross the planes of the secondary structure, it is easy to see that the susceptibility for the corresponding direction of the magnetic field will be structure sensitive, whereas for the other direction of the field the orbits will not cross the secondary structure, and so the susceptibility will not be structure sensitive.

Two Theoretical Problems

In addition to the experimental work of Goetz at Caltech, Zwicky has been active in dealing with the theoretical aspects of the problem.

There are two main problems requiring theoretical discussion: (1) to account for the existence of the secondary structure, and (2) given the secondary structure, to show why it should affect the properties of the material in the way it does. Zwicky has been concerned mostly with the first problem, to account for the origin of the structure. His arguments are all dedicated to the thesis that a perfectly regular lattice, either of similar atoms as in a metal, or of molecules as in NaCl, is not the configuration of lowest potential energy, but that a combination of the ordinary lattice shown by X-rays with a superposed secondary structure of much larger scale has smaller potential energy than the unmodified simple lattice. The crystal is supposed, in accordance with the principles of mechanics, to take up automatically the configuration of minimum energy. The calculations are difficult; the argument has taken several different forms, and I do not believe that it can be said even yet to be in its final form. The latest idea of Zwicky is that the relative instability of the simple lattice is connected with the free surface, and that a secondary mosaic structure works its way into the interior from the surface. Considerations are given showing that the order of magnitude of the secondary structure to be expected is the same as that observed. It seems to me that one of the weak points of the theory so far developed is that it is concerned only with the absolutely pure metal, whereas in practice it is known that the secondary structure is intimately connected with slight impurities, and it cannot even be claimed that the existence of a secondary structure has been established experimentally in the absence of slight impurities. At first it might be thought that the regularity of the secondary structure, as shown by Goetz's photographs, can hardly be consistent with an effect of impurity. But there are other phenomena which make this not impossible. For example, the Liesegang rings, which are deposited in gelatin when a solution of a silver salt diffuses into gelatin which has been impregnated with $K_2Cr_2O_7$, have a perfectly definite spacing. It is not difficult to imagine that the deposition of impurities from the metal as it solidifies may be similar to this process.

Complexities Masked by Impurities

We now consider the second group of phenomena, showing that great complexities are possible in the behavior of perfectly pure metals. Here the complexities are of such a character that they are smoothed over and obliterated by the presence of exceedingly slight amounts of impurity. This is somewhat paradoxical and the reverse of the usual state of affairs, for the presence of impurity often makes behavior more complex; as, for example, the sharp freezing of a perfectly pure substance is changed into freezing over a range with a continual change in the composition of the solid separating out, when impurity is added. The

experiments to which I refer were made by Schubnikow and DeHaas in the Cryogenic Laboratory at Leiden on the behavior of single crystals of bismuth at low temperatures, and in particular on the effect of a magnetic field on the resistance. The necessity for great care in the purification of bismuth was first shown by the great irregularity and nonreproducibility of the resistance at liquid hydrogen temperatures of bismuth from different sources. For example, the ratio of the resistance at 20° abs. to that at 273° abs. of the purest spectroscopic bismuth available with a total impurity of about 0.006 per cent was 0.11; the corresponding figure for the purest bismuth obtainable from commercial sources was 0.53, whereas another grade of bismuth frequently used in the construction of instruments gave the ratio 1.21. The spectroscopic bismuth was now purified still further by three different procedures. The first was simply repeated recrystallization into single crystal form from the melt, it being well known that many impurities are segregated during solidification. The second and third methods were chemical methods of resolution and reprecipitation. The result of these further purifications was to reduce the ratio 0.11 still further to approximately 0.05, and this ratio was now the same and reproducible for material prepared by all three methods, indicating that the purification had been carried as far as necessary, at least from the point of view of this phenomenon. It is interesting that the residual resistance of the highly purified bismuth at helium temperatures proved to be of the same order of magnitude as that of other pure metals, such, for example, as gold, indicating no essential difference in this phenomenon between other metals and bismuth, although bismuth with an exceedingly slight amount of impurity does act like an essentially different substance.

The effect produced by a magnetic field on the electrical resistance of this ultra pure bismuth was now measured at liquid helium temperature. It should be mentioned that in these experiments the crystal axis was parallel to the direction of current flow, and the magnetic field was perpendicular to it. Definite and reproducible values were obtained, which are doubtless characteristic of pure bismuth, with no extraneous effects, although similar measurements on ordinary bismuth are very far from reproducible. There are many surprising features about the results. Perhaps the most spectacular result of all was the enormous magnitude of the effect, a field of 30,000 Gauss increasing the resistance by a factor of almost 150,000 at liquid helium temperature. Another surprising result was that the effect varies greatly, sometimes by a factor of nearly 3, depending on the orientation of the *secondary* crystal axis to the magnetic field. (The secondary axis has no connection with the secondary structure discussed above.) This means that if the rod is rotated about its length in a magnetic field perpendicular to the length the resistance may vary by as much as a factor of 3, depending on the

orientation. But to me the most surprising result of all was that the effect does not increase smoothly with the magnetic field, unlike all other known magnetic phenomena, but the dependence is far from simple, with at least half a dozen points of inflection in the curve between 0 and 30,000 Gauss. To express the complete relation between magnetic field and orientation with respect to the secondary axis, seven terms in the expansion in Fourier's series are necessary, and each of the coefficients is itself a highly complicated function of the field.

So far as I know, no attempt has yet been made to find a theoretical explanation of these magnetic phenomena, and I shall certainly not attempt it. The point which I want to emphasize is that this is a very striking example of great complexity in the behavior of a pure substance, which may be entirely altered in appearance and obscured by the presence of amounts of impurity so small as to be beyond treatment by ordinary methods.

Complexities Connected with Quantum Effects

The third group of phenomena revealing unsuspected complexities in the behavior of ordinary substances are definitely known to be connected with quantum effects. It is now beginning to appear that there are a great many phenomena in this class; most of these are important only at low temperatures, where we are perhaps not surprised to find them, but it is also becoming evident that important phenomena of this character may be expected under more ordinary conditions. Perhaps the most definite and striking suggestion of what may be expected here goes back to the discovery by Simon in 1922 of certain anomalies in the specific heat of NH_4Cl. Since that time, Simon, with various collaborators, has shown that in the first place there is a similar anomaly in the specific heat of NH_4Br, and secondly that connected with the specific heat anomaly there is also an anomaly in the thermal expansion. The specific heat anomaly consists in a gradual rise of the specific heat above that to be normally expected. This rise first becomes just perceptible at about 150° abs., but then grows rapidly until at 243° abs. (about −30° C.) the specific heat is about twice the value to be expected. At this temperature it drops in a range of a few degrees back to its normal value. Fig. 1 shows the nature of the effect. The anomaly in the specific heat of NH_4Br is very similar, and the maximum departure from normal occurs at very nearly the same temperature. NH_4I also has

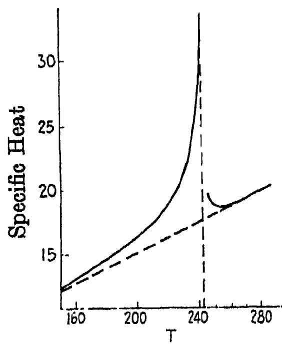

FIG. 1.—MOLECULAR SPECIFIC HEAT OF NH_4Cl PLOTTED AGAINST ABSOLUTE TEMPERATURE. [*From paper by F. Simon: Ann. Phys.* (1922) 68, 241.]

a similar effect, but at a slightly lower temperature. The volume anomaly of NH_4Cl is similar in character. This may best be studied by measuring the thermal expansion, which in the same way as the specific heat increases from its normal value, at first slowly, and then rapidly, until at the same temperature as the maximum anomaly in the specific heat it is of the order of 15 times the normal value; from this point it drops back in a few degrees to normal. The net result of the anomaly is that above the temperature of the anomaly the volume is somewhat greater than would normally be expected. The changes of

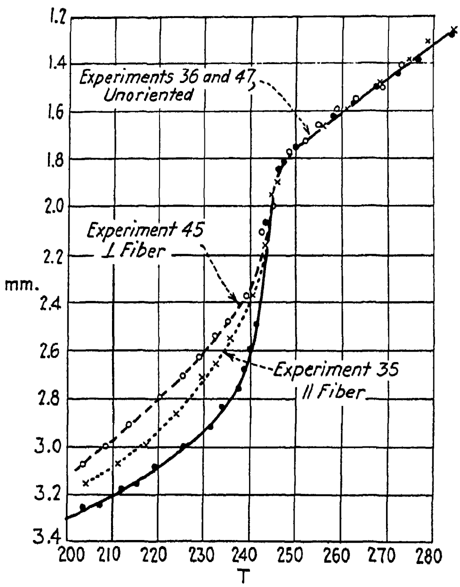

FIG. 2.—CHANGE OF LENGTH OF NH_4Cl PLOTTED AGAINST ABSOLUTE TEMPERATURE [*From paper by F. Simon and R. Bergmann: Ztsch phys. Chem.* (1930) **8,** 255.]

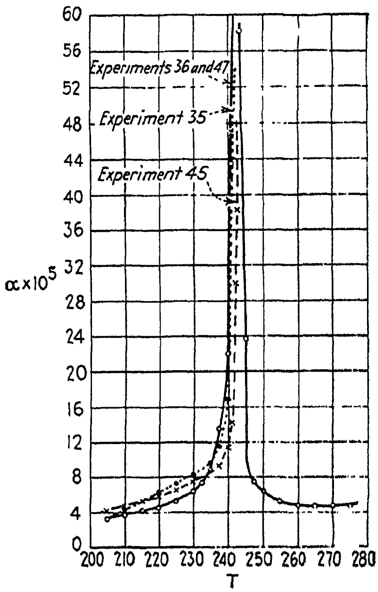

FIG. 3.—THERMAL EXPANSION OF NH_4Cl PLOTTED AGAINST ABSOLUTE TEMPERATURE. ORDINATES OF THIS CURVE ARE SLOPES OF CURVE OF FIG. 2. (*From the same paper as Fig.* 2.)

length and the thermal expansion are shown in Figs. 2 and 3. The volume anomaly of NH_4Br is similar in some respects and different in others. There is similarity in that the abnormal behavior is spread over a range of temperature, rising to a maximum at the same temperature as the maximum of the specific heat anomaly, and then dropping back rapidly to normal, but the behavior is different in that the anomaly is of the opposite sign, the thermal expansion being negative in the region of the anomaly, with the net result that the volume is less above the temperature of the anomaly than would be normally expected.

A combination of a volume effect with a thermal effect is of course exactly what occurs during a polymorphic transition, and it is at first

natural to seek the explanation in this way. The outstanding difference between this and an ordinary polymorphic transition is that instead of being sharply discontinuous the anomaly is spread over a range of temperature. Such effects occur, however, in polymorphic transitions in the presence of impurities unequally dissolved in the two phases, and the explanation might be sought here. However, an X-ray determination of the lattice structure made by Simon above and below the temperature of anomaly shows no change whatever in the crystal lattice, so that this possibility must be discarded, and we are evidently confronted with something quite different from the ordinary polymorphic transition. It may be mentioned that both NH_4Cl and NH_4Br do have ordinary polymorphic transitions, which are sharp, as is to be expected; these occur at temperatures between 100° and 200° C., 200° or so above the range of the anomaly so that there can be no confusion between the two phenomena. NH_4I, however, does have the corresponding polymorphic transition at a temperature of $-18°$ C., which is so near the temperature of the new anomaly that they could easily be confused, so that Simon did not make as exhaustive an examination of NH_4I as of the two others.

It is well known that the temperature of an ordinary polymorphic transition is displaced by the application of hydrostatic pressure, just as the temperature of melting or vaporization is displaced, and furthermore, the magnitude of the displacement involves the volume difference and the latent heat between the two phases by the well-known formula of Clapeyron. Now this new anomaly involves also volume differences and differences of total heat, the distinction being that the effects are now spread over a range of temperature instead of being sharply discontinuous. Analogy leads one to expect that the temperature range in which the anomalies occur might be similarly displaced by the action of hydrostatic pressure. Thermodynamic analysis does show that there is a precise parallelism between the two cases and that an equation the exact analogue of Clapeyron's equation governs the displacement by pressure of the temperature range in which the anomaly is situated. In the case of the ordinary transition the temperature is raised by pressure if the phase stable at the higher temperature has a larger volume than the phase stable at the lower temperature, which is normal for most substances, but the transition temperature is depressed by pressure if the low-temperature phase has the larger volume, as in the case of abnormal substances like water and ice. Applied to our examples, this means that the temperature of the anomalous region of NH_4Cl should be raised by pressure, because NH_4Cl expands on passing from the low-temperature to the high-temperature form through the region of anomaly, but that on the other hand the temperature of the anomaly of NH_4Br should be displaced by pressure to lower temperatures, because NH_4Br contracts on passing through the anomaly from lower to higher temperatures.

Effect of Pressure on Temperature

Recently, I carried out experiments to test these predictions, and have obtained results exactly as expected. The effect was first found with NH_4Cl, where one expects the temperature of the anomaly to be raised by pressure. At 0° C., 30° above the temperature of the maximum of the specific heat anomaly, the length was measured as a function of pressure, and an abrupt break was found in the curve at a pressure of 3370 kg. per sq. cm. Passing through the break the compressibility becomes abnormally high, and then after an interval of 2500 kg. per sq. cm. or so, assumes again its normal value. At the still higher temperature of

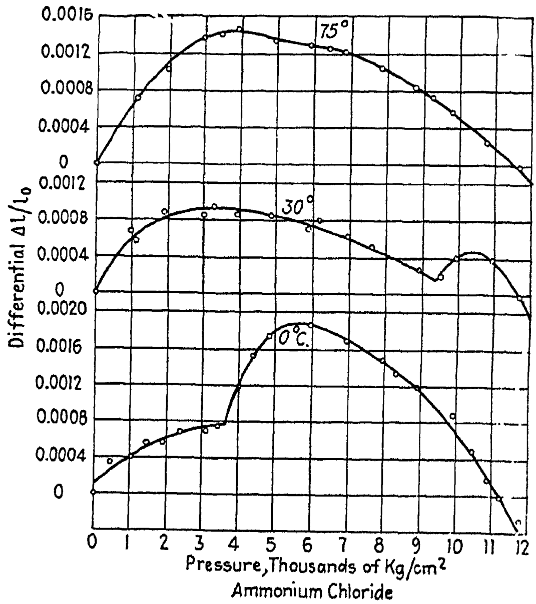

FIG. 4.—EFFECT OF PRESSURE IN DISPLACING TO HIGHER TEMPERATURES THE ANOMALY OF NH_4Cl.

In order to magnify the effect the ordinates are not the actual changes of length, but the difference between the actual change and the change that would have taken place if the relation with pressure had been linear.

30° C. a break was also found, but now at 9390 kg. per sq. cm. a considerably higher pressure, as is to be expected, because the temperature and pressure rise together. The behavior at 30° is much like that at 0°, except that the discontinuity is less in amount, and is not spread over so wide a pressure range. A third set of measurements at 75° disclosed no break of this kind up to a pressure of 12,000 kg. per sq. cm., the range of the apparatus, which is what would be expected, because extrapolation of the curve drawn through the points at −30°, 0°, and 30° indicates that at 75° a considerably higher pressure than 12,000 would be necessary to produce the effect. The results are indicated in Fig. 4, in which

are plotted the differences between the actually observed changes of length and the changes which would have been observed if the relation had been linear. It is to be particularly noticed that the relation between length and pressure is completely reversible, and that there are no hysteresis or lag effects, the experimental points lying on the same smooth curve whether they are obtained with increasing or decreasing pressure. Although the curve at 75° does not show the same anomaly as do those at 0° and 30°, it does nevertheless show a minor anomaly in the region between 4000 and 7000 kg. per sq. cm., which seems to be definitely beyond experimental error, and is doubtless another sort of complication than that now under discussion.

Turning now to NH_4Br, we expect pressure to depress the temperature of the anomaly to below $-30°$. This was first checked by measurements

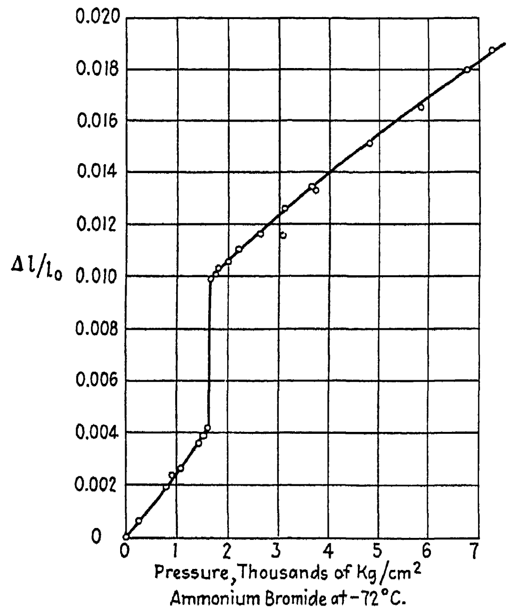

FIG. 5.—EFFECT OF PRESSURE IN DISPLACING TO LOWER TEMPERATURES THE ANOMALY OF NH_4Br.
The effect is much larger than for NH_4Cl and is sufficiently well shown by plotting the total fractional change of length against pressure.

of the volume as a function of pressure at 0° and 75° C. At both temperatures perfectly smooth curves were obtained, with no discontinuity of any kind, as was to be expected. Measurements were then made at $-72°$, the temperature of mixed solid CO_2 and alcohol. It is fortunate that the commercial demand for solid CO_2 which has sprung up in the last years makes it possible to obtain this substance in sufficient quantity and economically enough to make it feasible to cool the rather heavy pressure cylinder to as low a temperature as $-72°$ C. One hundred pounds of CO_2

and 4 gal. of alcohol proved to be sufficient. Another technical difficulty was in obtaining a liquid suitable to transmit pressure, practically all ordinary liquids being frozen solid by the pressure at so low a temperature. Compressed gases, which would not freeze, have the disadvantage that large volumes are necessary, and furthermore are dangerous. Fortunately it proved that iso-pentane is capable at this temperature of supporting a pressure of more than 7000 kg. per sq. cm. without freezing, and this proved to be ample. In Fig. 5 the results are shown which were obtained for the change of length at $-72°$ as a function of pressure. There is, as was to be expected, a discontinuity in the length, and therefore also in the volume; this occurs at about 1630 kg. per centimeter. The discontinuity is now so sharp as to be indistinguishable from the ordinary polymorphic transition. The only effect at this temperature corresponding to the range of temperature over which the anomaly is spread at atmospheric pressure is the anomalous upward curvature of the graph below the pressure of discontinuity.

It is possible to go further with the information that may be obtained from these curves and to calculate, for example, the total heat absorbed in the anomalous range. For NH_4Cl it turns out that this is approximately independent of pressure and temperature, and is about 100 cal. per mol. The total heat of NH_4Br is also not greatly different from this. However, although these questions are interesting for themselves, they are not immediately concerned with our main thesis, and I do not discuss them further.

Explanation of Effect of Absorption of Heat

The most interesting question here is the explanation of the existence of the effect. Evidently some sort of internal change must be taking place in the material. What sort of change is there that can be responsible for the absorption of an amount of heat of the same order of magnitude as that of an ordinary polymorphic transition and at the same time be consistent with the experimental fact that there is no change in the space lattice? Doubtless the most important feature in the situation is correctly reproduced in the explanation offered by Pauling. The fundamental idea of this explanation is that at low temperatures the NH_4 radical is more or less tightly restrained by its neighbors, so that in addition to the ordinary vibrational to and fro motion of its center of mass, it can only oscillate back and forth through a restricted range of angle. A crude picture of the situation would be that the hydrogen atoms project like prongs from the heavy nitrogen atom in their center, and because of their entanglement with their neighbors prevent angular oscillations of more than a certain amplitude. As the substance expands with rising temperature, however, the prongs eventually get free enough so that the radical may rotate continuously instead of only oscillating. The temperature range during which the radical is acquiring freedom to make

complete revolutions is the range of the anomaly. How does this picture explain the specific heat phenomena? According to the classical pre-quantum mechanics, this would have been no explanation at all, because the energy of a degree of freedom is a fixed quantity, so that in passing from oscillational to rotational motion, since there was no change in the number of degrees of freedom, there would be no change in the energy of the motion, and so no thermal effects. According to the quantum mechanics, however, the energy of a degree of freedom has a unique value only at high temperatures, and at a low temperature a degree of freedom may have less than this energy. The most familiar manifestation of this is the falling off of the specific heat curves of ordinary substances from their constant value at ordinary temperatures to zero at 0° abs., as shown in Fig. 6. The temperature at which the falling off begins depends on the tightness with which the vibrating system is bound in position; the smaller this binding force, the lower the temperature. Thus in Fig. 6, curve 1 corresponds to a degree of freedom with relatively tight binding, and curve 2 to one with relatively loose binding. At the temperature τ in the diagram the degree of freedom corresponding to the curve 2 has absorbed in warming from 0° abs. a total amount of heat equal to the area under the curve 2, while the degree of freedom corresponding to 1 has absorbed a considerably smaller amount of heat; namely, the area under the curve 1. In spite, however, of the difference in the total amounts of heat absorbed, the specific heats at the temperature corresponding to the curves 1 and 2 are the same (same ordinate of the two curves).

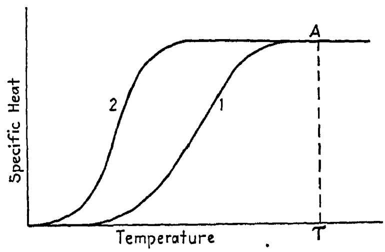

FIG. 6.—DIFFERENCE IN SPECIFIC HEAT CURVES FOR A SUBSTANCE WITH TIGHTLY BOUND ATOMS (1) COMPARED WITH ONE WITH MORE LOOSELY BOUND ATOMS (2).

The application to our problem is now obvious. In the position in which only oscillational motions are possible, the binding forces are relatively tight, and the heat curve corresponds to curve 1, whereas when rotational motion becomes possible, the binding forces become less and curve 2 describes the relations. In the temperature domain in which we are passing from oscillational to rotational motion we are passing from curve 1 to curve 2 and must absorb a total amount of heat equal to the difference of the areas under the two curves. It is this extra amount of heat absorbed during the transition, which is entirely absent from the classical picture, which accounts for the abnormally high specific heat. When the transition from oscillational to rotational has been completed we return again to the normal specific heat corresponding to the common ordinate A on the two curves 1 and 2.

I believe that you all will feel that this explanation is very beautiful, and I think we must consider that it is essentially correct. The same sort of ideas are also competent to explain other anomalies of some of the metals and of hydrogen gas at low temperatures, and so they acquire enhanced probability. I think it is clear, however, that this simple explanation will have to be supplemented in some respects, for in its present form it offers no explanation of the striking difference in the volume changes of NH_4Cl and NH_4Br which accompany the anomaly, one an expansion and the other a contraction.

Anomalies in Thermal and Volume Effects

There is no reason to think that there may not be other examples of anomalies in thermal and volume effects spread over a range instead of being sharply located, for mechanisms similar to that suggested by Pauling must be of fairly frequent occurrence, particularly in complicated compounds. There are, in fact, a large number of effects already known which doubtless come under this or some similar category. Thus, it has been very recently discovered at the Geophysical Laboratory in Washington, by Kracek, that $NaNO_3$ has thermal and volume anomalies very much like those of NH_4Cl, except that the region of the anomaly is in the neighborhood of 250° C. instead of at −30°. The explanation may well be similar to that of Pauling for NH_4Cl; namely, that the NO_3 radical, which is similar in its structure to the NH_4 radical, passes from oscillatory to rotational motion. The moment of inertia of the NO_3 group is so much greater than that of the NH_4 group that it is to be expected that the corresponding temperature would be much higher, and this agrees with experiment. Another example is NH_4F, studied also by Simon, who found again an anomaly, although very much less pronounced than that for the chloride. An interesting negative example is that of $N(CH_3)_4Cl$. This is similar in structure to NH_4Cl, the H being replaced by the CH_3 group, and the two compounds are very closely related chemically. The moment of inertia of the $N(CH_3)_4$ radical is, however, so much higher than that of the NH_4 radical that it is to be expected that very much higher temperatures would be required to produce corresponding effects; in fact, it may easily be that the necessary temperature would be above the temperature of decomposition. I made a search for the existence of the anomaly at room temperature up to a pressure of 12,000 kg. per sq. cm., but with negative results, as would be expected.

The possibility of transitions of this kind evidently opens the whole problem of polymorphic transitions. It is not at all inconceivable that the major part of the region throughout which the anomaly is situated should be so narrow as to simulate, under ordinary conditions of temperature and pressure, the sharpness of an ordinary polymorphic transition.

A very strong suggestion in this direction is contained in the observation above, that the transition of NH_4Br when running under pressure at $-72°$ is very much sharper than when taking place at atmospheric pressure and $-30°$. With so many possibilities as are now being recognized in the way of transitions, it is evident that the completest possible study should be made of the properties of the substance above and below the transition range, and, in particular, a study should be made of the lattice structure by means of X-rays. It is not at all impossible that some of the new transitions which I have found at high pressures are of this new quantum type. For example, the transition of metallic cerium is like that of NH_4Br in that the graph of volume against pressure has the same abnormal upward curvature below the transition pressure, which

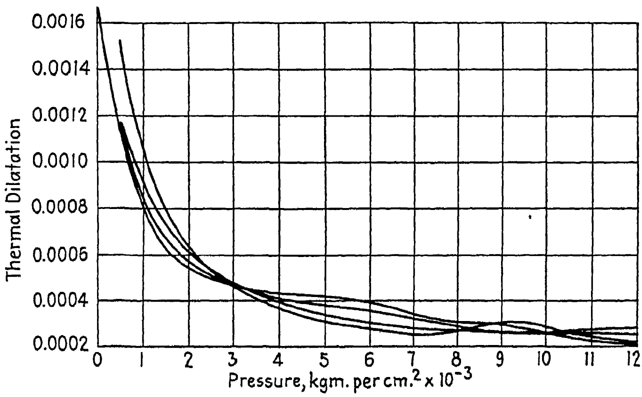

FIG. 7.—ANOMALIES IN THERMAL EXPANSION OF ETHER AT VARIOUS TEMPERATURES PRODUCED BY PRESSURE.

At pressures below 2000 kg. per sq. cm. the order of the curves reading from the bottom up is 20°, 40°, 60°, 80°. Crossing of curves or a curve rising with increasing pressure indicates an anomaly.

means that the compressibility increases with increasing pressure, instead of decreasing, as for the vast majority of substances. Another transition, hitherto only realized under pressure, which may be of this type is that of metallic cadmium, the chief argument being merely that it is so different in some of its properties from an ordinary polymorphic transition. There is no reason why anomalies of this sort should be confined to solids, but one might expect very complicated possibilities in organic liquids of complicated compounds, especially now that it is recognized on X-ray evidence that structures do exist in the liquid not greatly unlike crystal structures. There do not seem to be many such anomalies that can be definitely identified under ordinary conditions of temperature increase at atmospheric pressure, but if pressure is added to the temperature variable, giving two degrees of freedom, and increasing the range in the same ratio that a plane area is greater than a line, it is evident that the possibilities might be enormously increased. This does

in fact correspond to what I have found experimentally. Practically all the organic liquids examined with sufficient precision show ranges in which their behavior is anomalous; the anomaly may consist in a compressibility increasing with increasing pressure or falling with increasing temperature, or a thermal expansion increasing with increasing pressure. Fig. 7 shows one example of such behavior. Corresponding to these volume anomalies, there are also anomalies in the specific heats; these, however, at present can be detected only indirectly by thermodynamic methods from the volume anomalies, and not by direct measurement. I have no doubt that the explanation of effects of this sort, when finally arrived at, will be found to be in many respects like that of Pauling for the anomalies of NH_4Cl.

The clear appreciation that the simple materials of ordinary experience may exhibit under ordinary conditions the various kinds of complexity in their behavior which have been discussed will evidently have a far-reaching effect on our theoretical ideas of the structure of ordinary matter. The complete working out of such theoretical implications, however, is for the future, and I shall content myself with the suggestions already made. But there is another direction in which the recognition of the widespread existence of such complexities will affect our actions, and this is in the recording and reduction of our observational data. It is usually accepted as axiomatic that scattering observations which do not fall on a smooth curve are probably incorrect, and that only smooth curves are to be employed in reporting experimental data. In fact, apparently there has been growing up among physicists a sort of sixth intuitive sense as to what sort of smoothness a physical curve is most likely to have, and various people interested in the discussion of errors of observation have been concerned to get some mathematical formulation of what the experimental physicist does in drawing free hand the curve which seems to him best. The mathematical suggestions have been most varied in character, ranging from simple suggestions demanding the continuity of perhaps the third derivative to more complicated suggestions demanding the maximizing or minimizing of perhaps the entire area under the curve of the second derivative. The value of all such intuitive and instinctive demands is obviously much lessened with the recognition that anomalies may be expected which are truly characteristic of the material, and which become more accentuated as the purity of the material becomes greater. These possibilities evidently make the task of the experimental physicist much more difficult. He can no longer blithely discard a discordant observation as probably due to experimental error and so as of no significance; neither must he go to the opposite extreme of keeping in his final result all discordant individual measurements, because the chances of observational and instrumental error have not been at all reduced by the recognition of the possible existence of real

anomalies. Every discordant observation, when greater than the normal recognized instrumental or observational error, must be treated with more respect than hitherto; observations must be multiplied in the neighborhood of the discordant point to find whether the effect repeats and is in so far real. If the irregularity does repeat, one must make sure that it cannot be explained by an instrumental imperfection. If there are left unexplained irregularities in the data which one has not opportunity or time to clear up, these must be recorded meticulously in the final report of the experiment, in order that other experimenters may know exactly what the situation is. But this must all be done with a fine discretion, fortified by an understanding of the theoretical possibilities, or else the experimental papers of the future will be cluttered with the announcement of possible but unproved revolutionary new effects.

Mining Methods and Costs at Presidio Mine of The American Metal Co. of Texas

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(El Paso Meeting, October, 1930)

The Presidio mine of The American Metal Co. is situated in the "Big Bend" region, some 45 miles over state highway south of Marfa, a station on the Southern Pacific R.R. It lies 20 miles by road north of the Mexican border, in semidesert surroundings typical of this general section of the United States and the adjoining part of Mexico. All supplies are hauled in by truck from Marfa under contract. The concentrates and precipitates produced by the mill are carried by the same trucks to Marfa, from whence they are shipped, respectively, to the El Paso and Selby plants.

History

The mine was discovered about the year 1880 and was soon thereafter taken over for development by some army officers stationed at Fort Davis, Texas. It was later operated by the Presidio Mining Co., which treated the ore in a small pan-amalgamation mill from 1883 until 1913, when a cyanide plant was erected which has been in continuous operation ever since. When the present company, a subsidiary of The American Metal Co., Ltd., took over the mine in April, 1926, the ore-bearing zone had been worked for about 2200 ft. to the east of the outcrops down the dip of the formation, at which point it is at a vertical depth of 500 ft. below the surface. Up to that time the mine had produced some 1,150,000 tons of ore averaging about 17 oz. silver.

After a development campaign the company started milling again in February, 1927. Since then the production has been approximately 4700 tons per month of ore; of which the yearly average grade has varied from 19.7 to 23.2 oz. silver, $0.20 to $0.50 gold, and 2.0 to 2.5 per cent. lead.

No other property in this district has ever produced tonnage of any importance.

Geology

The area covers a series of sedimentary rocks, principally limestones' shales and sandstones, with a few beds of conglomerate. Of these, the

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lowest formation seen on the property is a limestone, of Permian age, locally called the Cibolo limestone. In one of the shafts over 400 ft. of this limestone is exposed. The lower part is thin-bedded to shaly, dark in color, and is several hundred feet thick; above, it grades into thicker bedded dark gray limestone; the upper 200 ft. is faintly bedded to unbedded and is light gray in color. This upper part of the Cibolo formation is the ore-bearing rock of the mine. It is separated from the overlying Lower Cretaceous rocks, principally bedded limestones, shales and sandstones, by an unconformity. Overlying the Cretaceous rocks on the neighboring hills is a series of eruptive flows.

The sedimentary rocks have been folded into a rather gentle dome and in the vicinity of the main workings dip to southeast or south at from 6° to 20°, with an average of about 12°.

The whole area is extensively fractured and faulted. Many of these fissures contain intruded irregular dikes of an acid to intermediate fine-grained igneous rock, which is post-mineral in age. The sedimentary rocks in many parts of the mine have been intruded by small sills along bedding planes, and along the unconformity at the top of the Cibolo limestone. The most important dikes include one crossing the center of the mineralized area in a northeasterly direction, with a width of several feet; a second narrow dike striking to the northwest, crossing part of the main ore area; and a third, some 15 ft. wide, striking nearly east, which bounds on the north the known area containing commercial orebodies in the old part of the mine.

As far as can be ascertained, all of the movement in the faults has been post-mineral in age, and in many cases where they cross an orebody the fault planes contain drag ore. The most important fault is known as the Mina Grande. It strikes nearly north-south and at the time the present operating company took over the mine it bounded on the west the mineralized zone which had been worked with profit. It has an average downthrow of approximately 250 ft. in its west wall. Many other faults with throws of usually less than 100 ft. cross the orebodies with varying strikes. Of these the most important to the present operators was one with an upward displacement of 40 ft. in its southwest wall, which had cut off the southwestern extension of the Tranquelino orebody. Both this fault and the Mina Grande fault zone are shown on the plan and sections of the western part of the mine workings (Figs. 1 and 2). At about 1000 ft. west of the Mina Grande fault, there is a large fault with several hundred feet of throw, and beyond it are several others with varying displacements; these, however, are at some distance from where any orebodies are being worked at present.

The orebodies are rather flat mantos which lie at several horizons in the Cibolo limestone, in general parallel to the bedding of that rock. They are typical replacement bodies and, as can be seen on Fig. 1,

they are very irregular in trend. The ore is siliceous and consists largely of quartz, with some calcite, dolomite and iron oxide, containing silver as chloride and as the sulfide argentite. Spots of argentiferous galena, and occasionally lead carbonate and sulfate, occur through the ore. Where the bodies contain much iron oxide the ore is comparatively soft; such ore is not continuous and is mixed with harder sections of siliceous ore, or with beds of unreplaced limestone, or blocks of coarsely

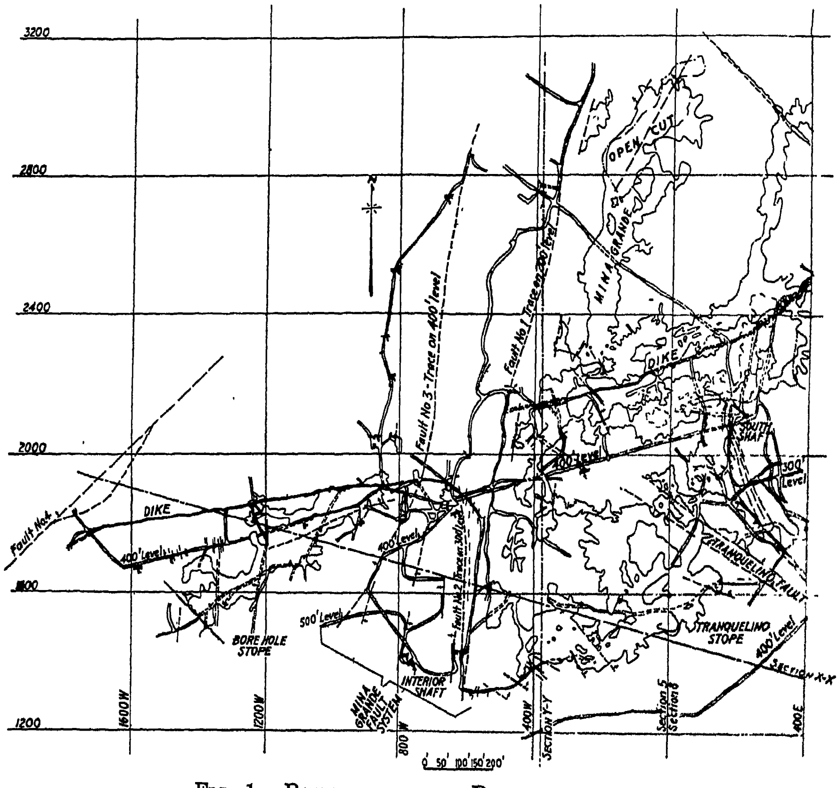

FIG. 1.—PLAN OF PART OF PRESIDIO MINE.

crystallized calcite. The limestone foot and hanging walls are usually irregular. On the sides of the bodies there are no sharp walls, the mineralization gradually grading out into silicified honeycombed limestone, and then into unreplaced, comparatively unaltered, country rock. The limestone in the vicinity of the orebodies is generally finely recrystallized and occasionally shows spots of pinkish color and some dendritic manganese stain.

The principal ore deposits developed by the present operating company are the Bore Hole and Tranquelino bodies. The former was developed by driving in the favorable limestone horizon in the practically

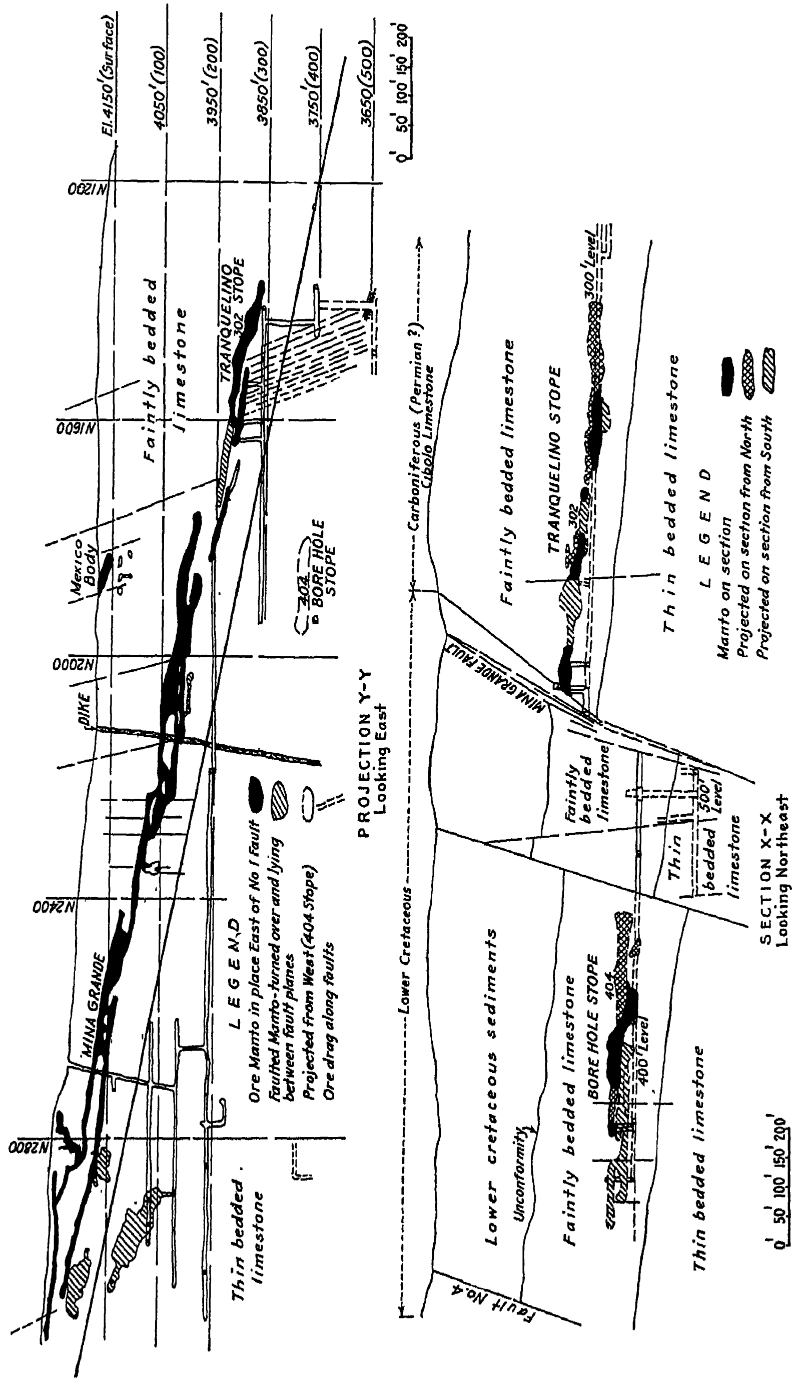

FIG. 2.—PROJECTION Y-Y AND SECTION X-X OF FIG. 1.

virgin country west of the Mina Grande fault. The Tranquelino is the extension of an old body of the same name, southwest of a fault up to which it had been mined many years ago. The latter ore has now been developed westward to the Mina Grande fault, and prospecting is being carried on to find its further extension in the downthrown block to the west. Thus, the principal present mining operations are concentrated in a section south and west of the old southwestern part of the mine, at a vertical depth of from 200 to 500 ft. below the surface. It is felt that the source of a large part of the mineralization of the old mine lies to the west or the southwest, and accordingly it is believed that further important extensions will be developed in that direction.

Characteristics of Orebodies and Enclosing Rocks

The character of the ore and wall rock has been covered under the last heading. The ore, though soft in spots, stands up well and no timbering is required to support it during development. The hanging wall, though only dipping ordinarily about 12°, is of firm, uniform, thick-bedded limestone. It needs no support, even over large stopes.

Attention has already been called to the fact that the bodies are comparatively flat, with considerable local irregularity in both foot and hanging walls. Fig. 1 shows how irregular the orebodies are in plan, and how difficult it is to predict the trend of probable extensions, since only in a few cases does there appear to be any really definite control by a fissure, or fracture zone, for any considerable distance. This irregularity is accentuated by the fact that many sections of the bodies are too low in grade to mine, and this necessitates leaving pillars of varying size, mining being carried out around them until extensions of ore of better grade are found. All pillars shown in the stopes on the plan were left, not for support to the back, but because of their low grade.

The sizes of the stopes vary considerably as the bodies are followed. Locally they may widen out laterally along some zone of fissures, or for some obscure cause, and it can be seen that the two main bodies reach horizontal widths, in places, of around 300 ft. The thickness (height) also varies from place to place, and in one body may be, along its margins, only from 2 to 6 ft., while in the center or at points where it shows a tendency to make up, locally, into a higher horizon in the limestone, it may reach a thickness of 40 ft.; 10 to 15 ft. might be taken as the average heights of the stopes being worked at present. In general, it can be said that the cross-section of commercial ore in these bodies varies considerably more in a short distance than does that of the typical replacement body of oxidized lead ore as developed in the important limestone districts in the northern part of Mexico; also, that the trends of the Presidio bodies are even more irregular and hard to predict over any

considerable distance; and that mining is made more difficult by the absence of relatively clean limestone walls, such as usually occur in the Mexican lead replacement bodies, along the lateral limits of the ore.

So far there has been no variation in the character of the ore with depth, and the deepest parts of the mine contain essentially oxidized ore such as existed at the outcrop. It has already been indicated that the values are erratically distributed through the ore, and that many sections of the bodies develop with a grade that is not marginal under present conditions; on the other hand, some parts of the stopes yield ore of much above average grade in silver, probably due to concentrations from leaching and reprecipitation during the oxidation of the deposits.

In addition to the major faults already mentioned, there are many minor faults and slips, which have a few feet of throw, as well as many open fissures of varying strikes, often as much as 1 ft. wide, and water courses. In the stopes these fissures tend to confuse the extensions of bodies, cause additional irregularities in them, break up the ore with consequent lowering in grade, and thus add to the difficulties of stoping.

Methods of Prospecting and Exploration

Although all of the upper 200 ft. of the Cibolo limestone apparently is favorable for ore deposition, the principal bodies have occurred in the lower part of this section, and at the present time prospecting is being practically confined to the search for new bodies, and for extensions of those formerly mined, in this lower part of the favorable horizon. Whereever possible the prospect drifts and crosscuts are driven at an elevation that would be somewhat below any probable ore occurrence, not only to make these workings available for ore extraction but to take advantage of the discovery of secondary iron oxide, which may have leached down into fissures below any orebody. In looking for extensions the probable future trends are predicted as closely as possible, but the irregular nature of the deposits often causes a body to make an unexpected turn, which renders useless some crosscut that was driven in the anticipated direction. Accordingly it has been found impracticable to drive too far ahead of the known position of any body, and expense is saved by prospecting for extensions only short distances in advance.

At all times a reasonable amount of exploration is carried out in the most favorable horizon, in the hope of picking up new bodies not directly connected with those now known. Roughly, it can be said that the present operating company has developed about 9.0 tons of ore per foot of exploration and development work.

Diamond drilling, as an exploration method, has been tried but found to be unsatisfactory, owing to the number of open fissures and to the broken-up nature of much of the ground by the extensive minor

faulting and fracturing. The water was usually lost before a hole had been driven far, and lost bits caused an excessive carbon cost. It is possible that diamond drilling might be used as a method of prospecting with short holes from stopes in the search for near-by branches, or for overlying or underlying mantos, especially in the less fractured blocks of ground, without excessive difficulty or cost, and will probably be tried on such work at some time in the future.

Sampling and Estimating Tonnage

The description of the irregularity of the mineralization indicates how difficult it is to block out or estimate tonnage far in advance. Formerly an effort was made to trace the extensions of the bodies for some distance, and to sill them out at definite intervals to determine cross-section and grade, but on account of the irregular nature of the trends, walls and distributions of values, it was found that this entailed excessive cost, caused much unproductive work and was not worth while. It has now been decided that the ore cannot be blocked out indefinitely ahead of stoping requirements, although effort is made to open up the ground in advance by drift and raises for a reasonable distance. Pilot faces are carried in the stopes to ascertain trends as far as possible.

Once a certain body has been mined for some distance, it is possible to estimate what it is likely to produce over a given length of extension. It is also found that its average grade is fairly uniform wherever opened. It is thus possible to make a rough estimate as to tonnage and grade to be expected over a section that has been indicated by several scattered raises.

Development

The mine has two main working shafts. The first, called the South shaft, was sunk in the western part of the mine in the neighborhood of the outcrops, and is that through which most of the ore is now being hoisted; it is 400 ft. deep, with principal levels at intervals of 100 ft. At present most of the work is being done on the third and fourth levels. The fourth level opened up the new country west of the Mina Grande fault, and will continue to be the main tramming level in the western part of the mine; from it an interior shaft has been sunk to 500 ft. and a fifth level is being driven to prospect for any branch from the Bore Hole body on the dip of the beds to the south. Fig. 1 shows the relation of these workings.

The second shaft from the surface, the East shaft, lies 1850 ft. north-east from the South shaft, and is 700 ft. deep. It and the eastern part of the mine workings are not shown on the plan. The upper terminal of the aerial tram connecting the mine with the mill, which lies in a valley one mile to the east, is near the collar of the East shaft, and is

connected to the South shaft by surface narrow-gage tracks over which 2-ton side-dump cars are hauled by gasoline locomotives.

Shafts and Winzes.—The interior shaft from the fourth to the fifth level was recently completed. This is the only shaft or important winze which has been sunk in recent years, so that it is not worth while to go into details or costs under this subject.

Drifts and Cross-cuts.—All drifts and cross-cuts are driven approximately 5 by 7 ft., on contract. No timber is required. Light Leyner-type drifters are employed, with standard column and arm, using 1-in. hollow round steel. The usual round is of "toe-cut" type, with 18 holes. The crew consists of a contractor, helper and one or two mucker-trammers, depending on distance from the shaft. The explosive is 40 per cent. gelatine powder.

Raises.—Raises in limestone or ore are driven 5 by 6 ft., also on contract. No timber is required, with the exception of staging, and access is gained to the back by rope or cable ladder. Hand-rotated stopers, with ⅞-in. hexagon steel, are employed. The usual round uses a toe-cut with about 15 holes. For short raises the driller can handle the work, but after a height of 30 ft. is reached a helper is also used; one to two mucker-trammers, depending on distance to shaft, are needed. The explosive is 40 per cent. gelatine powder.

Open-stope Mining

Inasmuch as the limestone back stands without support over an opening of any size, open stopes are used throughout the mine. The ore is mined selectively, in order to keep it above a minimum grade, and pillars are left when the ore becomes non-marginal. Rough sorting is done in the stopes, the waste being placed in parts where it will not interfere with operations.

The idealized plan and section of a typical stope (Fig. 3) gives an idea of the difficulty of developing and mining an orebody at this property, because of the highly irregular character of the deposit. An extraction drift is run below a body, which is then tapped by raises at approximately 50-ft. intervals. Where the level is at an excessive distance below the part of the stope being worked, a sublevel is driven, to which the ore is dropped and then trammed to a main raise from the haulage drift. Where a body is over 25 ft. above a level, it is not economical to space raises closer than at 50-ft. intervals; this ordinarily necessitates more or less wheelbarrow work in moving the ore from a stope face to the nearest raise. Thus the flat nature of the deposit entails a relatively high mucking cost per ton. Due to the irregularity of the ore, each part of a body is of necessity worked in accordance with the existing conditions, and no standard procedure can be laid down. Were it not for the advisability of mining the ore to a maximum grade, with corresponding comparatively

expensive selective mining, and increased necessity for careful supervision, which is more than justified by increased return, it would be possible to lower materially the existing mining cost. In other words, a considerable tonnage of material running 6 to 10 oz. silver exists in the bodies, and if it were economically possible to also mine ore of this grade, possibly lowering the average mine grade to around 14 or 15 oz., it would be possible to cut down present costs about 20 per cent.

To illustrate the advantage to be derived from careful selective mining, the 160,000 tons mined in three years of operation by the present company have averaged 22 oz. silver, as compared to an average grade of 11 oz. for the last 500,000 tons mined prior to 1926.

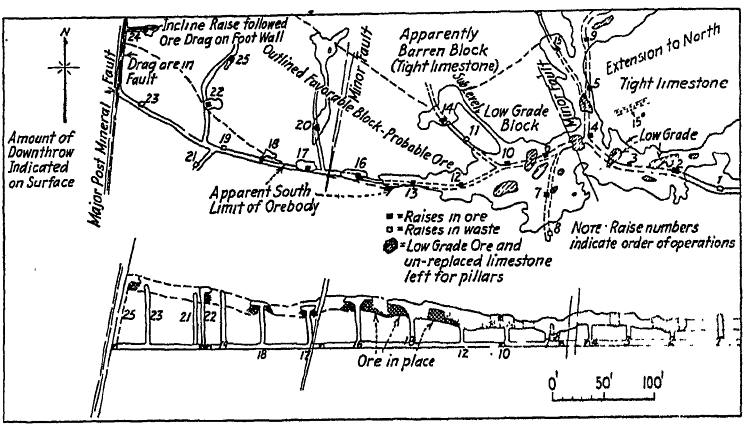

FIG. 3.—IDEALIZED PLAN AND LONGITUDINAL PROJECTION TO SHOW METHOD OF DEVELOPMENT.

Where the stope is over 8 to 10 ft. thick (high) ordinarily the section next the hanging wall is mined ahead, leaving a bench from which to drill; the bench is later broken as the cut is carried forward.

The use of scrapers has been given careful consideration and trial underground; but on account of the irregular nature of the stope floors, as well as the comparatively small size of individual working faces and the frequent changes in direction of the commercial ore, it does not appear that scrapers would reduce the cost of handling ore in the stopes sufficiently to justify the expenditure for equipment.

UNDERGROUND TRANSPORTATION

Ore is trammed by hand in 1-ton end-dump cars from the chutes in the extraction drifts to pockets at the South shaft. This shaft, prior to operations of the present company, was practically abandoned; work had been concentrated for some years in the eastern part of the workings around the two-compartment, well equipped East shaft.

The South shaft has been equipped to pull all ore and waste, hoisting being done with a small efficient electric hoist handling a 0.7-ton bucket without guides. This may appear to be rather a small installation to handle the tonnage, but it has proved capable of hoisting 250 tons of combined ore and waste on two shifts at a reasonable cost. Men are not handled in this shaft; they either walk down to work through the gently inclined stopes or are lowered through the East shaft.

Percentage of Extraction

Due to selective mining, less than 75 per cent. of a body is broken as ore, the remainder being left as pillars and along the sides of the stope. There is practically no dilution with waste from the walls; the waste and low-grade ore sorted out, both underground and on the surface, come from inclusions in the bodies.

It is estimated that 6 per cent. of the material broken is discarded underground as waste. Sorting at the shaft collar rejects as waste at the present time about 10 per cent. of the ore hoisted; this is a greater proportion than was formerly sorted out, and although it increases somewhat the cost per ton the expense is well justified.

Wages and Contract System

All work underground, excepting pipe work, track work, timbering and supervision, is done on a contract basis. Contracts are figured more or less on the basis of day's pay in the district, as follows: $4 for machinemen; $3 for muckers and trammers. The contractor is paid a fixed price per unit of work, and is charged for powder, caps, fuse and carbide. The men working for the contractor are paid by him through the company.

The prices paid on stope contracts vary greatly according to the conditions; they include delivery of ore into shaft pockets. An average price would be about $1.50 per ton.

The prices paid in limestone drifts, crosscuts and raises average $7 per foot; this also includes tramming of waste to shaft.

All work is measured and paid for twice each month. In some cases a bonus is given to the contractor for footage above a certain minimum. All underground labor is Mexican.

At times considerable "buscon" work is carried out in old stopes. This includes mining small bodies with a grade above average. Payment to the buscon contractor is made at a predetermined amount per ounce silver per ton by assay. Sometimes this is on a sliding scale to encourage mining the ore clean.

Ventilation

No artificial ventilation is necessary in any part of the mine.

Power Plant, Tramway and Crushing Plant

The power requirements of mine and mill are supplied by separate power plants. The mine plant consists of three De La Vergne oil engines, two of twin-cylinder and one of single-cylinder type, totaling 340 hp. rated capacity. These engines drive three air compressors, delivering a total of 1350 cu. ft. of free air per minute, a 90-kw., 440-volt a.c. generator, and several minor pieces of equipment. Power costs for 1929 averaged $0.0154 per horsepower-hour.

The aerial cableway connecting the mine to the mill is of the double-rope type, 1 mile long. Buckets hold 825 lb. of ore. Tramway costs are not included in the direct mine costs, which are given in Tables 1 to 3.

Before January, 1929, the coarse crushing of the ore was done above the upper tramway terminal bins. A new plant doing both the coarse and fine crushing is now in operation below the lower tramway terminal at the mill. Crusher costs, also, are not included in the direct mine costs.

Safety

Because of the relatively flat stopes and the substantial character of their walls, open stopes are employed with a very low accident rate. As far as available records go, there have been only three deaths due to underground accidents in the last 30 years of operation.

Costs

Tables 1, 2 and 3 show mine costs and efficiencies, and Table 4 gives total operating costs, including mine, tramway, crusher, mill and general charges.

Table 1.—*Summary of Costs during Year 1929 at Presidio Mine*

[Tons Ore Hoisted during Period, 54,524[a]

Work Performed	Underground Costs per Ton Ore Hoisted							
	Labor	Supervision	Compressed Air, Drills and Steel	Power Cost	Explosives	Timber	Other Supplies	Total
Development								
In ore	$0.026		$0.007		$0.008		$0.002	$0.043
In rock[b]	0.733		0.160		0.100		0.020	1.022
Mining	1.442	$0.146	0.430		0.177	$0.025	0.056	2.276
Transportation (underground)	0.435			$0.049			0.049	0.533
General underground expense[c]								
Total	2.636	0.146	0.597	0.049	0.294	0.025	0.127	3.874
Surface expense (directly applicable to underground operation)	0.077						0.050	0.127
Total	$2.713	$0.146	$0.597	$0.049	$0.294	$0.025	$0.177	$4.001

[a] Tonnage does not include waste sorted out on surface.

[b] Costs of development in rock include tramming of waste.

[c] Distributed under Development, Mining, and Transportation.

TABLE 2.—*Summary of Costs during* 1929 *at Presidio Mine in Units of Labor, Power and Supplies*[a]

Tons Ore Mined and Hoisted, 54,524

	Development	Mining (Stoping)	Total
Labor (man-hours per ton)			
Breaking (drilling and blasting)	0.41	1.28	1.69
Mucking	0.52	1.46	1.98
Tramming	0.26	0.74	1.00
Hoisting			0.37
Supervision			0.14
General			0.59
Total labor underground	1.19	3.48	5.77
Average tons per man per shift (underground)			1.39
Average tons per man shift on surface properly chargeable to underground operation			17.83
Average tons per man-shift (total mine)			1.29
Labor, per cent. of total cost	19.40	53.92	73.32
Power and Supplies			
Explosives (lb. per ton) 40 per cent. gelatin	0.82	1.18	2.00
Total power (hp-hr. per ton)	5.43	17.51	23.70
Air compression	4.78	14.99	19.77
Hoisting	0.65	2.52	3.17
Lighting			0.53
Miscellaneous			0.23
Other supplies in per cent. of total supplies and power	10.14	30.26	40.40
Power, supplies and sundries, per cent. of total cost	7.61	19.07	26.68
Per cent. of total cost	27.00	73.00	100.00

[a] All per-ton figures are based on total tonnage of ore mined and hoisted, not including waste sorted out on surface.

TABLE 3.—*Detail of Costs during* 1929 *at Presidio Mine in Units of Labor, Power and Supplies*

Raising and Crosscutting in Untimbered Excavation 5 by 7 Ft. in Uniform, Hard, Tough Limestone

Labor (man-hours per foot)	
Breaking (drilling and blasting)	3.35
Mucking	4.46
Tramming	2.23
Total labor	10.04
Feet per 8-hr. shift	2.39
Feet per man-shift	0.80
Power and supplies (per foot)	
Explosives (lb. per foot)	7.59
Total power (hp-hr. per foot)	50.40
Other supplies	$ 1.01
Labor, per cent. of total cost	72.13
Power, supplies, and sundries, per cent. of total cost	27.87

TABLE 4.—*Total Operating Costs at Presidio Mine, Including Tramway, Crusher, Mill, and General Charges*

	1927	1928	1929
Tonnage mined	48,429	57,441	54,524
Direct mine cost	\$3.57	\$3.50	\$4.00[b]
Direct tramway cost	0.17[b]	0.20[b]	0.11
Direct crusher cost	0.20	0.16	0.14[c]
Direct mill cost	1.92[c]	2.07[c]	1.86
General charges[a]	1.31	1.33	1.47
Total operating cost per ton	\$7.17	\$7.26	\$7.58

[a] General charges include, in addition to the ordinary general overhead, all insurance premiums, property taxes, proportion of head office expenses, legal expenses, and cost of smelter representation.

[b] Includes surface motor haulage.

[c] Includes fine crushing. New crushing plant in operation in 1929.

Top Slicing with Filling of Slices, as Used at the Charcas Unit of the Cia. Minera Asarco, S. A.

By Howard Willey,* El Paso, Texas

(El Paso Meeting, October, 1930)

Mining operations of the Charcas unit at present are limited to the Tiro General mine at Charcas, in the State of San Luis Potosi, Mexico. The Tiro General mine was first operated during the Spanish occupation of Mexico for rich silver ores in an oxidized zone along the outcrop of the principal vein. In depth the ore is a complex sulfide with widely varying proportions of sphalerite, argentiferous galena and chalcopyrite; and until recent years, it was economically impossible to carry on mining otherwise than irregularly in sections of the veins richest in galena and chalcopyrite. The advent of selective flotation brought the orebody as a whole within the field of economic interest, and a flotation mill of 650 metric tons daily capacity was constructed in the year 1925. The mining of partly worked-out sections of the mine, as well as the very heavy ground in most of the stopes opened in virgin ore, presented an unusually difficult problem. This paper describes the so-called "filled top slice" stoping method finally adopted as the standard system.

Physical and Geological Conditions

The principal orebody of the mine is an oreshoot about 1400 ft. long in a fissure vein striking about east and west and with an average dip of 70° to the north. The footwall of the orebody is fairly clean-cut and regular both longitudinally and vertically and generally shows a well-marked post-mineral slip. The footwall rock varies from a fairly strong limestone to a broken, metamorphosed limestone which requires heavy timbering. In some sections it is an altered, crumbly porphyry, which swells badly and necessitates frequent renewal of drift and raise timber. The hanging wall is irregular and frequently an assay wall. The stoping width ranges from a few feet up to more than 90 ft.; it averages about 30 ft. For the most part the hanging-wall rock is a shattered and altered monzonite porphyry, nearly aways requiring heavy timbering.

The orebody is greatly fractured by post-mineral movement, which usually followed the footwall and left it as a smooth surface frequently covered with a slippery clay gouge. Two well marked transverse faults

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with wide crush zones have been recognized and there is evidence of several other transverse faults of small displacement. The most characteristic fracturing of the orebody is in planes striking parallel to the vein and varying in dip from practically horizontal to about perpendicular to the footwall. This fracturing is probably related to the post-mineral movement along the footwall and causes the whole orebody to be blocky, treacherous ground for stoping. Except in the extreme eastern and western portions of the vein, heavy timbering is required for all development work in the orebody.

Former Mining Methods

Until recent years, operations were confined to the opening of levels in the vein at 100-ft. intervals and mining the more leady portions of the vein by narrow cut and fill stopes. Fill was usually obtained by breaking lower grade or zincky ore from the sides of the stopes. Very little timber was used in stoping operations, and falls of ground and extensive caves were evidently frequent. Most of the stopes in the central portion of the mine were abandoned long before they neared completion. This mining was carried on to a depth of over 800 ft. below the outcrop of the vein.

More recently a central main waste pass was driven in the footwall and stoping was carried on in a systematic way by horizontal cut and fill with the use of stulls, timber cribs and stope sets. Caves and runs of ground were still frequent in spite of close timbering and filling.

With the building of the selective flotation mill and the consequent increase in the economic width of the vein as well as the possibility of profitably reopening and mining partly stoped sections and old fills, it became evident that for most of the mine cut and fill methods must be abandoned in favor of a more dependable method for heavy ground.

The 500, 600, 700, 800, 900 and 1000-ft. levels were the only levels open and operating at the time, and caving methods and top slicing were not advisable because of the certainty of caving the unopened upper section of the mine as well as part of the surface plant and probably the main hoisting shaft.

Therefore cut and fill work was changed to square setting with round timber in most of the stopes, which brought a great improvement in extraction and regularity of production. Even with square setting, stoping work was still difficult and dangerous in many sections of the mine. In the wider portions of the central section it was necessary to limit stope widths in some cases to panels three sets wide, with back-filling of all sets as quickly as possible to prevent their collapse. Top spiling frequently was necessary and, in many cases, side spiling as well. In the stoping of the old partly worked-out sections, open ground was

often encountered and the mining of the surrounding ore was extremely dangerous and costly.

Filled Top Slice Stoping Method

During a visit at El Bordo mine of the Santa Gertrudis Co. at Pachuca, the writer was favorably impressed with a top-slice method with the filling of slices, which was being used in the mining of old fills in sections of the mine where the avoiding of surface subsidence was a serious factor. A trial of the method at Charcas gave such favorable results that it was soon adopted as the standard method, not only for heavy virgin ground but also for the stoping of old partly worked-out sections and caved stopes.

Perhaps the method is best classified as "underhand cut and fill." Its advantages apply only to orebodies of which the walls must be supported by filling, and in which the support of fill overhead is easier and less costly than the provision of support for the ore and walls by an overhand method.

For some time the operators hesitated to adopt the method in the upper levels, which were known to have a large amount of broken ground and partly caved open workings, as they feared that the collapse of an open working might result in the dropping of a stope floor and the burying of workmen. It is quite evident, however, that the collapse of open workings is much more likely to result from the undermining of their surrounding walls by square setting than from underhand work at their tops. The irregularity of their tops usually results in their early discovery by the breaking through of small holes rather than a large opening; and they may be filled immediately, before further progress is made on the stope. If the open working is small, it should be filled with broken ore from the stope. The ore used for filling is recovered, of course, in the mining of the floors of the next slice. If the open workings are large, they may be filled with waste, which in the mining of the floors of the next slice is thrown in with the fill for the floors.

For the conditions at Charcas, the method has many advantages over square setting, the principal of which are: (1) greater safety for the workmen, (2) better total recovery and cleaner mining, (3) lower cost per ton mined.

Development Work

As now used at Charcas, the method requires a footwall drift having a 10 to 12-ft. pillar between the side of the drift and the vein, and double-compartment footwall raises at intervals of 100 ft. along the strike of the vein. These raises are driven to leave a 6-ft. pillar between each raise and the vein. Due to minor irregularities in the dip and strike of the vein, it

is usually explored in advance of the driving of the footwall drift, by means of a pilot drift in the vein, from which crosscuts are driven at intervals of 100 ft. The footwall raises are driven near the crosscuts to insure accurate information for the securing of a pillar as near as possible to 6 ft.

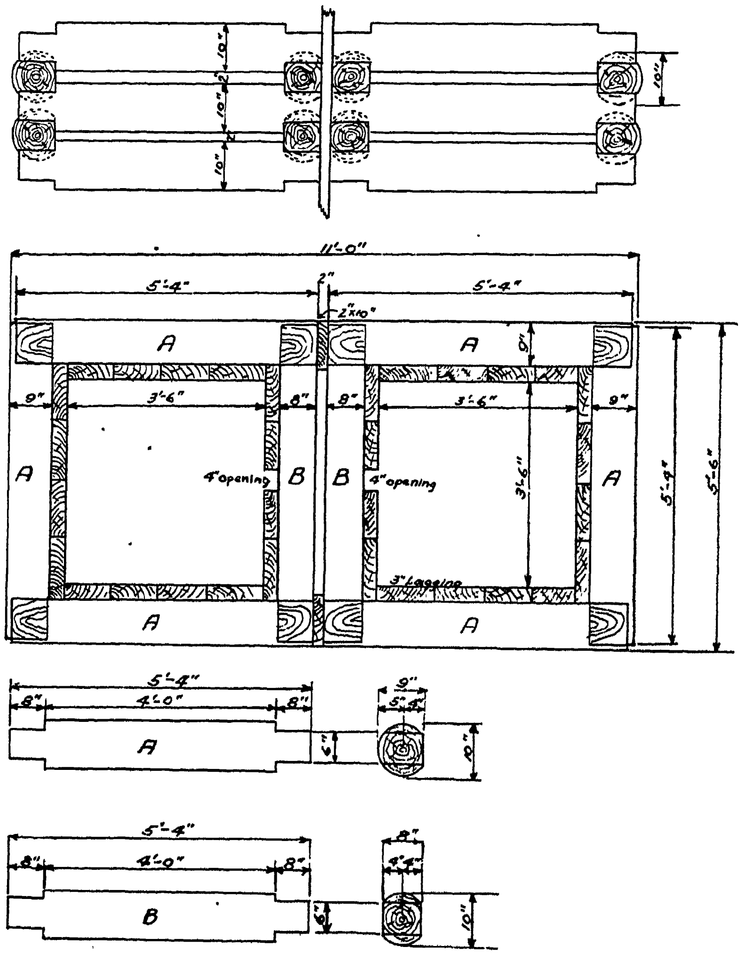

FIG. 1.—TIMBERING IN FOOTWALL ORE PASSES. STANDARD 10-IN. ROUND CREOSOTED CRIBBING.

in width between the side of each raise and the vein. These double-compartment raises serve as ore passes. Details of their timbering is shown in Fig. 1. This size of cribbing was adopted after a thorough trial of cribbing of smaller size and is necessary for satisfactory and uninterrupted operation with the comparatively long chute used, and the character and volume of ore passed through them. Timber treatment is not

necessary in the drier portions of the mine; but all cribbing for use in sections favorable to the growth of fungus is treated with creosote to a retention of 6 lb. per cubic foot.

Opening of Sill Floor

The sill floor of slice stopes is opened by regulation square setting on 5-ft. centers with native pine timber 8 in. dia. This square setting of the sill floor differs from regular square setting in that no sills are used under the posts.

As the square setting of the sill floor proceeds, sills are laid between the square-set posts from footwall to hanging wall. These sills are laid on 5-ft. centers and in a direction approximately at right angles to the strike of the vein. The sills consist of two 3 by 10-in. planks of native pine, laid double and with joints overlapped by at least 2 ft. The planks used are as long as can be handled conveniently, and no pieces shorter than 10 ft. are used in the bottom layer when it is possible to avoid it.

The sills are evenly supported with loose ore throughout their length and the space between them evenly leveled off to the tops of the sills with loose ore. A floor of 2-in. native pine plank is then laid on the sills. This flooring is most conveniently laid in 5-ft. lengths with the ends at the centers of the sills. Open spaces in the flooring at the square-set posts are most conveniently covered by short pieces of plank laid crosswise. The sawing of 5-ft. lengths from planks of random lengths will usually furnish about the necessary amount of short pieces for this work. Floor planks may also be cut away partly to fit around the posts, but care should be taken not to weaken a plank by cutting deeper than one-third of its width.

As flooring is laid, the square sets are backfilled as closely as possible with waste; care is taken to see that they are well blocked and that fill is placed close against all caps and girts. The closeness with which flooring and backfilling follow the advancing square sets is dependent upon the character of the ground; but it is a strict rule at Charcas that not more than two rows of square sets from footwall to hanging wall shall be opened without backfilling except when the vein is less than three sets wide. In other words, ground must not be opened to a width greater than that of two square sets in advance of the backfilling.

Slice Preparation

After the complete flooring and filling of the sill floor, a crosscut from the ore pass is started toward the vein at a vertical distance of 12 ft. below the sill floor. This crosscut is driven about 6 ft. 6 in. high by 5 ft. wide and is stopped after it has entered the vein about 6 ft. A narrow drift is then driven along the footwall of the vein for 50 ft. on either side of the crosscut. This drift is usually driven the full height of 12 ft. to the

floor above, and care must be taken to avoid opening more than the exact width necessary for standing two posts at 5-ft. centers under each sill. Lagging is used on both sides of the drift to prevent sloughing of the walls. This is very important, because any slabbing off of the walls removes support from the sills above and may result in their failure before the slice is completed. If the footwall is wet and slippery or the ore along the footwall is badly broken, the drift may be driven farther out in the vein in order to leave a pillar of ore along the footwall. In extreme cases, as when the ore has a tendency to swell, or when there is an unusual width of vein (50 ft. or more) that will require an unusual amount of time for the completion of a slice floor, it is advisable to drive this drift of the same size as regular development drifts and timber it with ordinary drift sets. These drift sets are salvaged as the mining of the slice proceeds.

It is very important, in any case, that no more than the necessary amount of ground shall be broken in the driving of these preparatory drifts, because they remain open throughout the mining of the slice.

Posts of 11-ft., 10-in. dia., native pine timber are used, with headblocks of 10 by 10-in. or 8 by 8-in. native pine lumber. The posts are tightened against the headblocks by wedges at the bottoms of the posts. Footboards or sills are not used.

Slicing and Filling

The order followed in the mining and filling of a slice is shown in Fig. 2. As soon as the preparation drift has been extended 50 ft. (10 sills) from the center of the ore-pass crosscut, ore breaking proceeds from footwall to hanging wall between the last two sills exposed by the preparation drift. The sills are supported, as the face advances, by 11-ft., 10-in. dia., native pine posts on 5-ft. centers with 10 by 10-in. native pine headblocks, in the same way as in the preparation drift. Details of the timbering are shown in Fig. 3. Light sprags cut from slabs or flooring are used between all posts near blasting and are left in place until the posts have taken sufficient weight to begin to cut into their headblocks.

As the opening of the first row nears completion, the breaking of ore from footwall to hanging wall is started between the sills second and third from the end of the stope, as shown at the right side in stage 2 of Fig. 2. Sills of two 3 by 10-in. planks are laid on 5-ft. centers from footwall to hanging wall and flooring of 2-in. plank is laid in the same way as for the sill floor. A small crosscut is then started into the footwall from between the two end sills, with its bottom a height of 6 ft. above the floor of the slice. Muck from this crosscut is piled on the completed flooring to a height of 6 ft.; it is held in place by gob lagging on the slice posts. Bamboo poles of 2-in. dia., spaced about 6 in. apart, are used for gob lagging, this being the cheapest material available for the purpose.

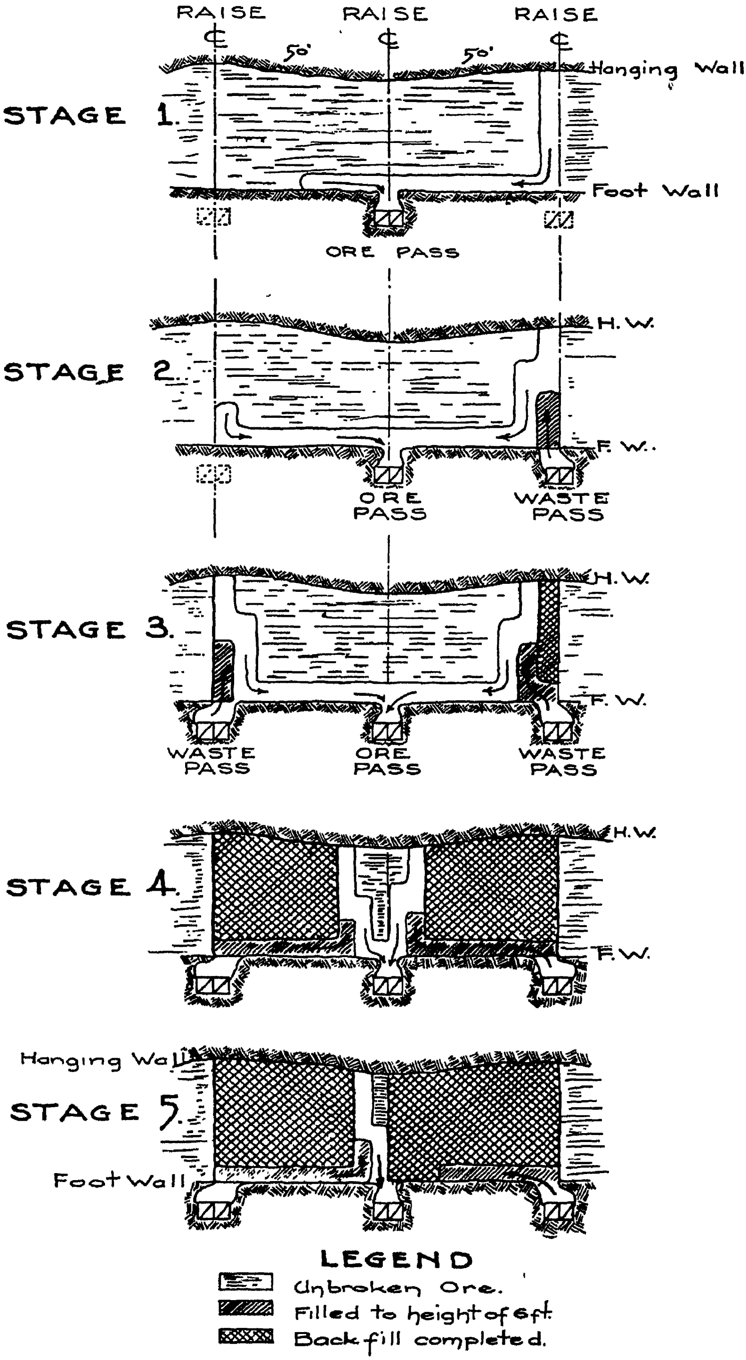

Fig. 2.—Plan of stope floor, showing progress of mining.

The crosscut at the end of the stope is advanced to the side of the footwall drift and widened along the side of the drift to allow for the placing of double-compartment cribbing of the size shown in Fig. 4. This cribbing is made from 8-in. dia. native pine and is treated with creosote (6 lb. per cu. ft.) when used under conditions where rotting is likely. The cribbing is installed not closer than 6 ft. to the footwall

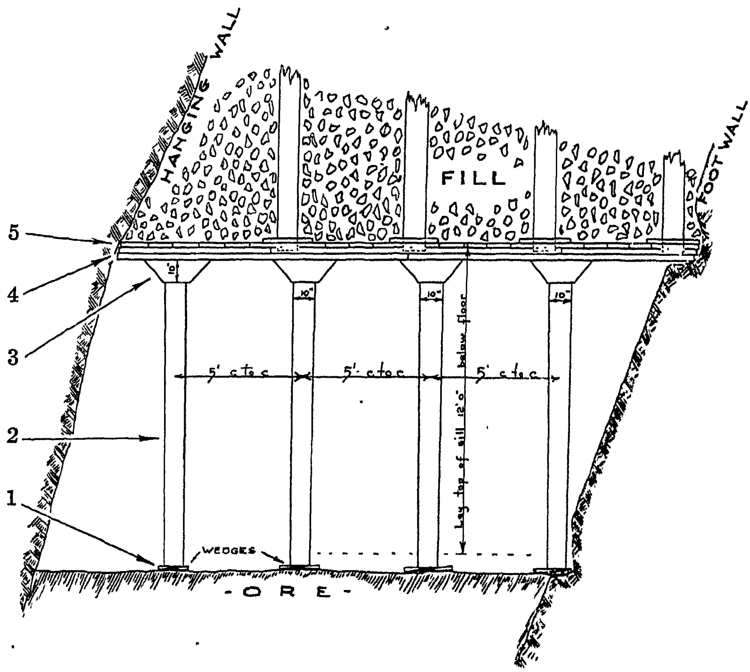

FIG. 3.—TIMBERING DETAILS. CROSS-SECTION OF STOPE.

1. Posts should be wedged at their lower ends to tighten them against their headblocks. Footboards are not necessary and the lower end of the post need not be cut square.

2. Posts are approximately 11 ft. long by 10 in. average diameter. The larger end of these posts should be up, in order to give as large a bearing area as possible to the headblocks and should be sawed square before lowering into the mine. Posts of 8-in. dia. may be used on the hanging wall side of the stope.

3. Headblocks are of sound 10 by 10-in. pine, 30 in. above, 12 in. below. Should be nailed to sill before posts are placed.

4. Sills are made of two 3 by 10-in. planks. Plank as long as can be lowered into stope should be used and ends must overlap about 4 ft. Sills must be laid exactly to 5-ft. centers.

5. Flooring is laid of 2-in. planks, cut 5 ft. long. Ends of flooring must be at centers of sills and supported by ore between sills. Gaps left at posts are covered by short pieces of plank about 24 in. long laid crosswise to flooring. *Posts must not rest on flooring or sills.*

of the vein and is supported on sills on the floor of the crosscut. A temporary chute of 2-in. plank is used for the convenient and rapid loading of wheelbarrows of 3-cu. ft. capacity. One compartment now serves as a manway and the other as a waste pass. In the meantime,

the procedure shown in Fig. 2 is being followed at the other end of the slice floor.

The mining of the slice floor now proceeds by the breaking of successive cuts 5 ft. wide, always beginning at the footwall and proceeding to

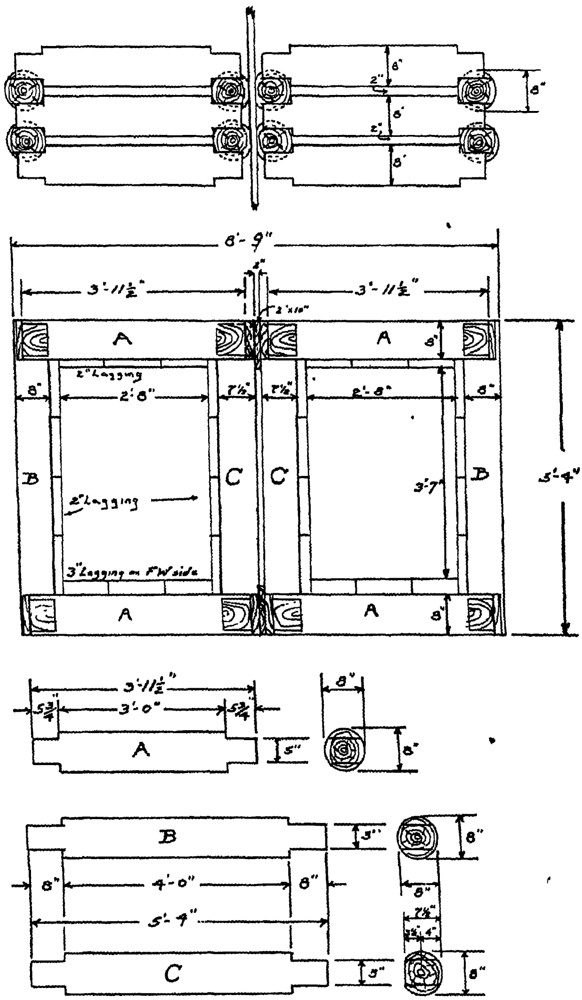

FIG. 4.—TIMBERING IN WASTE PASSES. STANDARD 8-IN. ROUND CREOSOTED CRIBBING.

the hanging wall. Filling to a height of 6 ft. follows behind as closely as can be done without interference with ore breaking. As soon as a row of posts has been filled to a height of 6 ft., backfilling tightly to the floor above begins at the hanging wall side and proceeds back toward the footwall. As the backfilling nears the footwall, a passage between the last

two rows of posts is lagged off with bamboo poles and left open along the footwall for the passage of wheelbarrows of waste filling from the footwall waste pass. Fig. 5 is a photograph of a typical stope, showing the passage along the footwall to the waste pass at the left and the advancing ore cut in the line ahead of the wheelbarrow in the foreground.

At the end of the stope near the waste pass, the space between the gob lagging and the unbroken ore of the adjacent stope block is filled with waste in order that no opening may be left which might allow caving or movement of the ore in the adjacent block.

FIG. 5.—TYPICAL STOPE.
Passage along footwall to waste pass at left; advancing ore cut in line ahead of wheelbarrow.

It is desirable that ore breaking and filling on one side of the ore pass shall be finished somewhat in advance of the work on the other side, as is indicated in Fig. 2. This will enable filling to be completed on one side of the ore pass and the preparation drift of the next slice started in under it before ore breaking is entirely completed on the other side.

The final stage of backfilling is in the passage along the footwall from ore pass to waste pass. This is started at the ore pass and retreats to the waste pass. The manway of the waste pass provides for the entry of men to carry out this work. This final stage of backfilling must be carefully watched, as it is a temptation to the workmen to tightly fill the end near the waste pass and conceal space left unfilled farther along the passage.

The mining of successive slices is carried on in exactly the same manner as for the first slice. Usually, however, there is considerably less weight on the timber of the second and succeeding slices than on that of the first slice. Figs. 6 and 7 are sections through the ore pass and waste pass of a typical filled top-slice stope and show the positions of the ore pass and waste pass crosscuts with respect to the slice floors.

A few of the more important details in the use of the method at Charcas are mentioned in the following paragraphs.

HEADBLOCKS

In nearly all sections where the method is used, the hanging wall is very heavy and there is a slight surface subsidence for a distance of over 1000 ft. from the outcrop of the vein, as shown by leveling over monuments. This movement of the hanging wall develops a pressure which

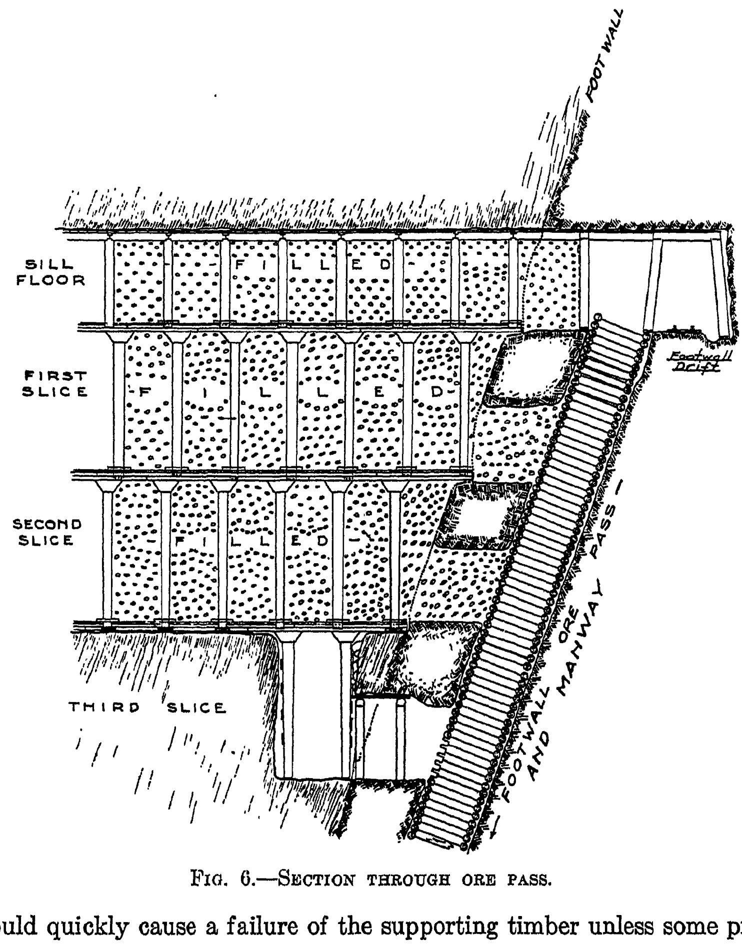

FIG. 6.—SECTION THROUGH ORE PASS.

would quickly cause a failure of the supporting timber unless some provision were made for cushioning the movement. The headblocks, by bearing in cross grain against the tops of the posts, are the weakest point in the timbering and provide for this movement by crushing down over the tops of the posts. The movement is slow and ordinarily the posts do not cut more than a short distance into the headblocks before filling can be placed and provide support for the floor. At times, however, headblocks are cut entirely through before filling is completed. It is evident, therefore, that the placing of the top of a post directly against a sill

would quickly result in the breaking of the sill and the collapse of the floor. For the same reasons, it is clear that the bottoms of posts must not be placed on the flooring or sills of a slice that is being mined.

Headblocks are somewhat similar in effect to the use of jacket sets in shaft work, and provide a cushion for the protection of the slice flooring and sills.

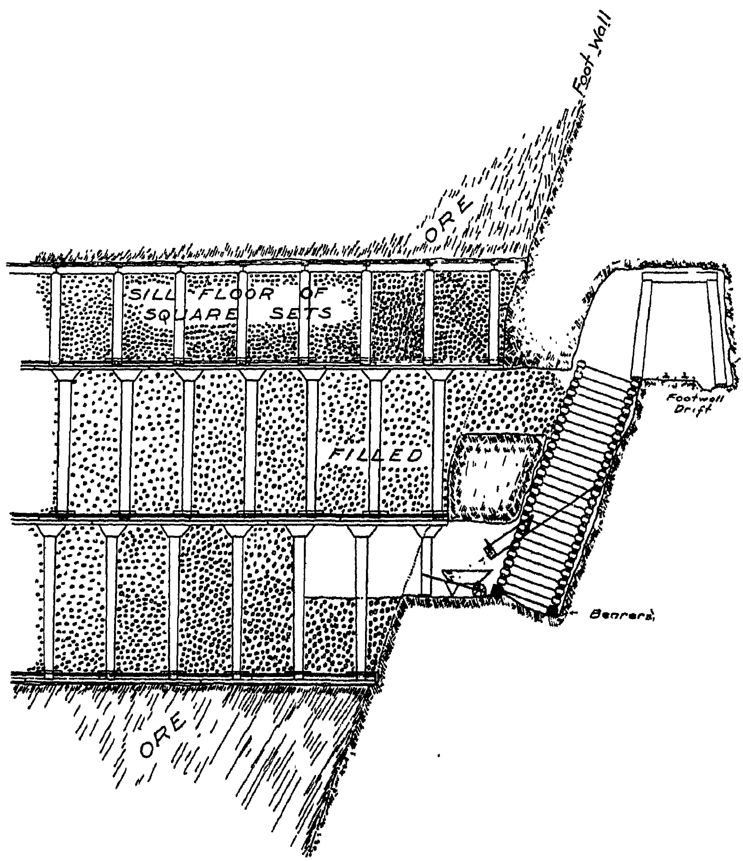

Fig. 7.—Section through waste pass.

Ore and Waste Passes

Two compartments with independent cribbing are an advantage for both ore passes and waste passes, as they allow all necessary repairs and timber renewals to be carried on in either compartment without interference with the other compartment and interruption to the stope work. In normal operation, the two compartments are used alternately, so that lining may be regularly inspected and repaired in each compart-

ment before it wears through and allows damage to the cribbing. At the time of opening a slice floor or the driving of a preparation drift in mixed ore and waste (as frequently happens in the upper levels), one compartment may be used for ore and the other for waste. Above the active slice floor, one compartment of the ore pass is provided with ladders and platforms and used as a manway and timber slide. The two compartments of the waste passes provide interchangeability for the carrying on of inspection and repair work, as well as a manway for emergency use and ventilation. The waste passes also serve the adjacent stope blocks as well as the stope shown in Fig. 2.

The maintaining of a raise or manway through the fill of slice floors is a dangerous practice, on account of the excessive load it is likely to carry by reason of the slight settlement of floors and fill and the consequent heavy spot which it causes in the mining floor. The principle of the method is to change all concentrated timber loads to a uniformly distributed load on the fill of each floor, and thus avoid cumulative concentrated loads. For the same reasons, timber cribs or rock pack walls should not be allowed on either the sill floor or any slice floor.

General Precautions

As production is dependent upon the number of working faces rather than upon the actual area of the stope block, particular attention must be given to the avoiding of any delays in the advance of a working face.

The backfill provides support for the flooring of the slice above and for the transfer of the load from the posts of the mining floor to the fill. As the mining of the next slice will remove their support from the bottoms of the posts, if the backfilling has not been well done, the posts will settle and allow movement of the fill above their tops. If this movement is sudden, there is danger of the collapse of the floor through sudden transfer of load and a run of fill may result. The carrying of filling close to the ore breaking, as indicated in Fig. 2, makes it easy to be sure that no openings are being left in the filling; it also gives early support to the timbering and thereby reduces the danger of the failure of a sill through the blasting down of posts or poor timbering.

Flooring and sills should be laid carefully and planks as long as possible should be used for the bottoms of the sills.

Practically all accidents to workmen in the use of the method have been due to neglect to bar down loose material along the sides of advancing faces or to allowing arches of ore to be left projecting out at their tops.

Costs

On account of the poor class of labor in the district and the unusually high cost of timber and lumber, actual cost figures are of no value for

comparison with stoping methods used in other districts. A direct cost comparison between square-set and filled top-slice stoping under average Charcas conditions shows a cost reduction as tabulated below in favor of the filled top-slice method:

	Mining Virgin Ore	Mining in Old Workings
Labor	$0.106	$0.102
Explosives	0.008	0.011
Lumber and timber	0.411	0.270
Total saving by use of filled top-slice stoping at Charcas	$0.525	$0.383

Miscellaneous

The production per man shift is approximately the same as for square-set work, average figures for the two methods under Charcas conditions being about as follows:

	Virgin Ore, Metric Tons	Old Workings, Metric Tons
Square set	1.55	1.24
Filled top slice	1.44	1.34

By working two shifts per day (not including Sundays) the average production for a stope block 100 ft. long, having a central footwall ore pass and a footwall waste pass at either end, is about 1000 metric tons per month. The recovery of ore by this method is practically 100 per cent. as against an estimated recovery of 95 per cent. by square setting with backfilling.

A most important advantage of the method is its easy reduction to a simple routine operation quickly learned by unskilled workmen.

DISCUSSION

(H. L. Carr presiding)

J. P. Hodgson, Bisbee, Ariz.—Why is it necessary to drive a footwall drift in the footwall on each level?

H. Willey.—In the mining of slices there is a certain amount of subsidence, and if we try to carry the drifts out in the vein and work underneath, of course there will be subsidence. Also, it is dangerous to leave openings overhead when working underneath, as the posts are liable to drop into the opening that is made and cause the drift to collapse.

J. P. Hodgson.—In other words, you believe that it is impossible to work the mine without footwall drifts?

H. Willey.—With the method described in my paper the footwall drift is essential.

J. P. Hodgson.—Another question—why do you adopt the filled top method? Is it because your plant is sitting on the mine and you cannot allow the ground to follow you down?

H. Willey.—It is partly that, and partly because at the time this method was adopted the upper section of the mine had not been opened. We knew there was a considerable amount of ore left there from old-time operations, but it was not opened sufficiently to allow us to get in and start operations. It was necessary to begin in the central section of the mine. If we were to use a caving method, naturally the workings up there would drop and we would lose the ore.

J. P. Hodgson.—So it was inadvisable to start stoping at the top of the orebody nearest the surface?

H. Willey.—Yes.

F. MacCoy, Berkeley, Calif. (written discussion).—Mr. Willey describes a method of mining adapted to a special condition, and it is a method that takes care of that condition very well. This condition is often found in re-mining old Mexican mines, where the high-grade pay streak has already been gouged out, leaving one or both walls of low grade, which by lower mining costs or newer methods may be mined with profit.

Sometimes the filling of the old stopes is worth mining with the "scab," but where it is of too low a grade to pay, the underhand square set method Mr. Willey describes may be the solution. Often these scabs are too small to stand the cost of the whole plan of development used at Charcas, but it is often possible to use a modified form, whereby a double compartment raise in the wall is used as manway, ore pass below the working floor, and waste pass of the part above. It is usually possible, by a little planning of each stope as a separate problem, to handle the work so that a slusher hoist can be used in moving the ore out and waste in for back-filling the square sets. The miner who is accustomed to use the slusher only in large cut and fill stopes may question this point, but the use of a little mechanical ingenuity usually solves the problem.

Drill Sampling and Interpretation of Sampling Results in the Copper Fields of Northern Rhodesia

By H. T. Matson* and G. Allan Wallis,* N'Dola, Northern Rhodesia, Africa

(New York Meeting, February, 1931)

In the Northern Rhodesia copper fields the size of the orebodies and the exceptionally consistent values over great distances made it possible to outline the ore with drill holes spaced at 1000-ft. intervals; in some cases, 2000-ft. intervals. Owing to the fact that the size of the deposits made it very evident that much outside capital would have to be called in to develop them and, owing to the wide spacing of the holes, it was essential that the sampling of drill holes be as accurate as possible, so that outside examining engineers could adopt the results as trustworthy within the limits of accuracy of the drilling method.

With this object in view, the field staffs of the Selection Trust group of companies have studied for the past three years, and are still studying, drill sampling in a very thorough manner, and for this reason it is felt that these notes may be of interest to engineers or students who are, or in the future may be, in charge of drilling operations.

The orebodies in this field lie in shales and sandstones, more or less indurated, in the lower strata of the Roan series, and the copper mineralization is in the form of sulfides predominantly, with some oxidized copper minerals. Those interested in the geology and history of the area can find full details in a recent paper by Anton Gray and Russell J. Parker.[1]

In some formations core recovery was good—between 90 and 100 per cent.—and where any difficulty occurred in recovering sludge, as in fissured ground, this was not insisted upon; but the samplers' difficulties, both practical and theoretical, do not commence until core recovery drops and sludge recovery becomes important, for if core is being ground up it is recoverable only as sludge, and naturally the sludge sample, which theoretically should contain only the material cut from the annular ring around the core, is being salted, in a sense, by the ground-up core. In the perfect core, where 100 per cent. core is recovered as well as 100 per cent. sludge, the two samples can be treated separately, as in the case

* Rhodesian Trust Selection, Ltd.

[1] A. Gray and R. J. Parker: Copper Deposits of Northern Rhodesia. *Eng. & Min. Jnl.* (1929) **128**, 372, 384, 429, 470.

of two channel samples cut side by side, but once core loss is considerable a mean has to be struck between the assay values of the core and sludge samples, and it is at once apparent that unless sludge recovery is complete, or nearly so, the sludge sample will be of little value.

Procedure on Drilling into Ore

Water Return

Upon striking ore in either shot drilling or diamond drilling the procedure is much the same. The hole is tested for water return, by pumping water down the drill rods, and its volume is measured, while at the same time the volume of water running over the collar of the hole is measured. Unless these volumes agree within 1 per cent., the hole is either cemented or cased, for a difference in the volume means that water is escaping into the country rock rather than rising to the surface. If the hole is caving seriously the same measures are adopted.

Drilling and Pumping Sludge

The water return being assured, a distance of not more than 4 ft. is drilled, while the water is pumped gently down the hole. Drilling is then stopped, the pump is put on at full pressure, and the sludge watched closely. Fine slime comes up first, followed by heavier sludge, which gets coarser until the water apparently clears. Pumping is continued, however, as it is found that there is considerable lag of the heavy minerals. Every three minutes a beaker is held under the sludge delivery pipe and the sediment examined until no more appears. It sometimes takes 45 min. or more for all the sludge to come up. Pumping is then stopped and the core pulled. The drill rods are measured carefully, which gives the correct distance drilled. In the case of a shot drill the sediment tube sludge is screened through a 10-mesh screen, and the +10 material is discarded while the −10 is added to the rest of the sludge. The +10 is discarded because it consists of shot and cavings, if any. In some instances reverse pumping of sludge has proved successful. This is done by screwing a stuffing box, which fits the rods tightly, into the top of the T piece at the casinghead. The pump is then connected to the T piece and the sludge is forced up through the rods. The advantage of this method is that the greater speed of water in the rods lifts the sludge more rapidly and reduces the amount of water which it is necessary to catch. In fissured ground, however, the ordinary method is better, as there is not so much lost in the fissures, owing to the smaller pressure.

Catching Sludge

So far the method of catching the sludge has not been considered. The sludge overflows from the horizontal arm of the casinghead T piece

and runs either in a pipe or launder to a point about 10 to 20 ft. from the hole. If a pipe is used it should be slotted along the top to allow the drill washout pipe to be inserted to wash down sludge that may have lodged in it. This launder, or pipe, discharges by gravity into a galvan-

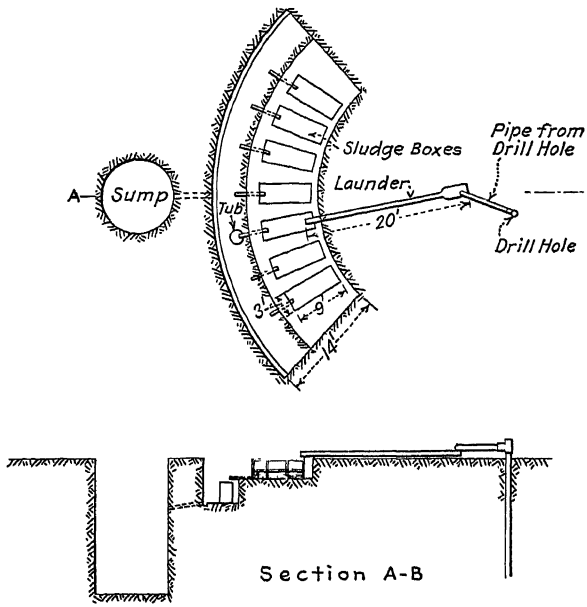

FIG. 1.—METHOD OF CATCHING SLUDGE.
A sump is used only when there is insufficient fall for natural drainage.

ized iron launder, which can be swung to discharge into any of the 10 or more sludge boxes arranged in a semicircle in a pit about 2 ft. deep (Fig. 1).

Fig. 2 shows a side elevation of a galvanized iron sludge box in a timber frame fitted with a decanting arm consisting of two elbows

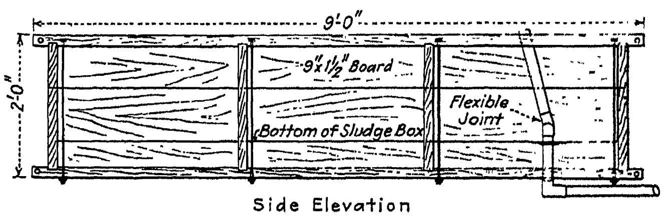

FIG. 2.—SIDE ELEVATION OF SLUDGE BOX.

connected by a nipple, which in turn screw on to a nipple in a flange sweated on to the bottom of the tank (Figs. 2 and 3).

As each box fills, the launder is swung over to the next, until the water no longer contains fine slime. As soon as the water is clear, the launder is swung over to an empty box, which is allowed to overflow when

full, as the heavy "lag" material settles easily. It is sometimes difficult to settle the slimes in the boxes, but a strong alum solution or, if this fails, a strong lime solution, is allowed to drip into the sludge launder and has the effect of hastening settlement enormously. As slimes settle out the decanting arms are pushed beneath the surface of the water and nearly all the water is drained off. Tubs to hold about 15 gal. are put under the discharge pipes and the decanting arm is screwed out of the flange in the bottom of the box. The sludge is then carefully washed out and dried over a slow fire. Caking and consequent burning is guarded against by stirring continually with a clean wooden paddle. Formerly the practice was to allow all the sludge to run into the top box of five or six boxes arranged in line, with overflow lips at the bottom end of each box. When all the boxes were full, samples of the overflow water from the last box were taken every 3 min., while the total quantity of overflow water was measured and the slime thus collected was dried, weighed and assayed. This method was discarded, as the flow of water from box to box kept the slime in suspension.

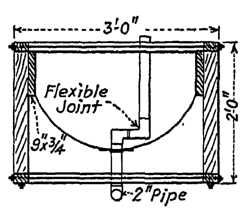

FIG. 3.—END ELEVATION OF SLUDGE BOX.

A method of catching the sludge which has been used with the diamond drills at one of the mines of another group, but with which the writers have had no practical experience, is interesting, and, the writers understand, satisfactory. A Challenge hand pump is used to draw water through a canvas filter sack contained in a cylindrical galvanized iron tank 20 in. in diameter and 3 ft. 4 in. high. Sludge from the drill hole is fed into the top of the filter bag. When a run is finished the bag is taken out, tagged, numbered and sent to a sample-drying shed, where it is hung in a current of hot air. When it is dry, the sludge is removed.

The disadvantages of the method would appear to be: (1) that the apparatus cannot be made readily in the field unless drilling is being carried on in some established mine or in some civilized locality; (2) a breakdown in the pump means either allowing the sludge to run away until the pump is repaired or holding up the drill; (3) a leak in the filter cloth might allow sludge to escape unnoticed unless a very close watch were kept on the pump discharge; (4) no provision for measurements of return water is made.

The sludge-box method, although appearing laborious, works well and cannot break down readily. Only standard pipe fittings are used. In the absence of a tinsmith, good wooden boxes can be made of pine shelving by a competent carpenter, and decanting devices can be replaced by a series of holes, one above the other in the box end, fitted with wooden plugs which can be removed successively from top to bottom as the sludge settles.

Sludge "Lag"

Before discussing core, there is one aspect of sludge recovery which should be brought out. In notes on drill sampling to be found in textbooks, lag of the sludge below the relative core is discussed. If this lag exists it can be attributed only to one or both of two causes; namely, incomplete cleaning of the hole, or a caving hole. A shot drill hole, for purposes of sludge recovery, can be regarded as an upward current classifier and the speed of the water at any point in the hole can be calculated. It remains, therefore, to consider only the heaviest particle the water must lift, and if this is recovered all lighter particles must also be recovered, always assuming that no loss of water is occurring. It has been found by experiment with a short core barrel that if shot is being carried high enough up the hole to be deposited into the sediment tube, the top of which is 10 ft. from the bottom of the hole, all sludge will also be lifted at least as high as the shot and therefore will nearly all be recovered, either with the shot in the sediment tube or with the water pumped out of the hole.

It has also been found that the ratio of percentage of water returned to surface to percentage of sludge recovered is not direct, but that the sludge recovery falls much more rapidly than the water recovery. This is due, no doubt, to the lag of the sludge in the upward current of water and to the preference of the sludge for coming to rest in a crevice through which water is escaping rather than for going up the hole, unless the upward speed of the water is great and the crevice small. From this can be evolved the rule that if the water return is complete, or very nearly so, and the sediment tube contains shot, sludge recovery is good. If water return is not good, steps should be taken to remedy the fault.

A caving hole can be detected by the presence of chips of rocks in the sediment tube. As soon as the quantity of these becomes great enough to make it apparent that the hole is caving of its own accord, and consequently will continue to do so after the rods are removed, the hole should be either cased or cemented and redrilled. A little caving will always be caused by the rods, but most of this is caught in the sediment tube, and will be discarded in the 10-mesh screen.

Experiments have shown that in a diamond-drill hole, as long as the water return is complete and the velocity of the water great enough, all sludge will be recovered if pumping is continued long enough, as already described. The rods should be free from oil, as this will collect the sulfides.

A low percentage of sludge recovery from a drill hole makes the sludge samples valueless, as it is then impossible to arrive at a true assay for the sludge, because the degree of classification effected by the upward water current on the recovered sludge cannot be more than guessed at.

Treatment of Sludge Samples

After a sludge sample has been dried it is weighed and crushed to −30 mesh and "quartered" down to two samples weighing about 4 lb. each. One of these is stored and the other goes to the crusher house. Here diamond-drill sludge is pulverized to −80 mesh and quartered down to about 1 lb. and sent for assay for copper. Between each quartering the sludge is rolled thoroughly on a square rubber sheet. The shot drill sludge, however, is treated as follows:

The sample as it comes from the rig is weighed and screened through a 60-mesh screen and the +60 material, which is a mixture of shot and sludge, is bucked until nearly all the sludge passes through the 60-mesh screen. The shot is removed from the remaining +60 material with a magnet, and is weighed. Any +60-mesh residue will be sludge and can be bucked to pass 60 mesh. The −60 material is quartered to about 1 lb. and sent to the assay office for copper and total iron determination.

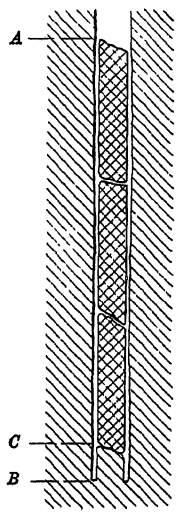

FIG. 4.—POSITION OF CORE AS IT IS BEING PULLED.

CORE SAMPLING

Pulling Core

Owing to the fact that a length of core frequently is left standing at the bottom of the hole when the core barrel is pulled, it is important that the distance from the bottom of the bit to the bottom of the core in the core barrel be measured before the core is removed from the core barrel. It is possible, of course, that this measured distance may not represent the length of core left in the hole, because the core may slip down the barrel when it is being pulled, but it is often near enough to the actual figure to be useful in ascertaining the probable correct position of the core relative to the sludge for a particular run. Fig. 4 illustrates this, where AB is the distance drilled, and therefore the section of the hole represented in the sludge sample, and AC is the length of core recovered, while BC is the length of core left standing in the hole, which should be included with the core AC when arriving at the combined core and sludge value for the section of the hole AB.

Laying Out and Measuring Core

When the core has been removed from the core barrel, the pieces are laid out carefully in the right order. It is well to number and indicate with an arrow in blue pencil the top and bottom of each piece, for it is easy to mix the pieces when examining them, especially when they are

small or do not fit well together, as a result of grinding. A sheet of galvanized corrugated iron makes an excellent table on which to lay out core. The distance apart of the top and bottom of the pull can be marked on the iron and the core can be laid out between these points as nearly as possible as it occurred in the hole. The core should then be divided into sample lengths, bearing in mind that it is better to have several short samples that are reliable in themselves than to have one or two samples composed of unreliable and reliable core mixed. Fig. 5 indicates the right and wrong way of dividing a pull into sample lengths.

The sample lengths should then be measured along the center line. The length is used only for purposes of reporting preliminary results and is not an accurate one where core recovery is poor. The length of hole represented by the sample should be measured also, but if core recovery is poor it is rather an arbitrary length unless the pull is treated as one sample. (This is the "length represented.") The totals of lengths represented equals the "pull length" or distance drilled between pulls.

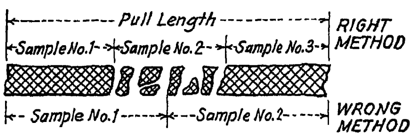

FIG. 5.—METHOD OF DIVIDING PULL INTO SAMPLES.

The diameter of the core should be measured carefully in several places for each sample. This is an important dimension, as on it is based the outside diameter of the hole, from which the percentage of sludge recovery is calculated.

The core for each sample length is weighed, first in air and then in water. This must be done carefully, because the specific gravity of each sample may be important.

The core is now ready to be split, but the geologist should examine the core before splitting, as well as afterwards.

Splitting Core and Crushing

The core should be split along the plane at right angles to the strike of the bedding in the core. It is well to chalk over one side of the core and leave the other side clean, as this makes it easier to see to which half of the core each individual piece belongs as the splitting goes on.

After splitting, half of the cores should be carefully weighed, crushed to pass 80 mesh and quartered. As large a sample as possible, up to 3 or 4 lb., is bagged, sealed and stored for reference, and a 1-lb. sample is sent for assay for copper; with a shot-drill core, assay is made for total iron also.

CALCULATIONS

Fig. 6 is a specimen sample ticket blank, one of which is filled in for each sample. The "description of losses" space is for describing any

breaks in the core—whether there is any evidence of grinding between the pieces and, if so, whether the material ground away was likely to be high, low or even grade. Core will often break at a vein of sulfide. The space above the "description of losses" is used by the sampler for a sketch of the core, indicating the point at which core loss has occurred.

SAMPLING SHOT/DIAMOND-DRILL REPORT

Mine.................... Date..............193...

Drill Hole No.... .... Sample No............... Samplers.................

Position in Pull........... ....

Depth

From.................. To...................... Dia. of Core..............

Pull Dia. of Hole.............

Length................. Length Recovered.........

Core Sample Length of Rods... ....

Length Represented..... Length Recovered. .... Length of Core Barrel.....

Weight in Air..........

Weight in Water........ Theoretical Volume....... Top of Rods to Datum....

Water Displaced........ Actual Volume...........

Specific Gravity.......... Weight of Sample........

Sample Recovery............ Pull Recovery..............

Description of Losses..

....

....

....

Formation..

....

Mineralization...

....

Sludge.............. Sample No....... Total Weight..... Water Return %...

Pumping— Straight/Reverse Sludge Recovery %.........

Remarks...

....

....

Sampler's Signature....................

FIG. 6.—SPECIMEN SAMPLE TICKET BLANK.

For example, two actual pulls will be considered, one from a diamond drill and one from a shot drill, there being two samples to the pull in each case. Table 1 gives the figures.

TABLE 1.—*Data from Two Pulls*

Ref. No.		Diamond-drill Pull			Shot-drill Pull		
		Sample 243	Both Samples	Sample 244	Sample 26	Both Samples	Sample 27
1	Pull, lb., from		697.62			596.55	
2	Footage, to		703.95			599.65	
3	Pull length, ft		6.33			3.10	
4	Sample length, ft	3.20		3.05	1.53		1.50
5	Represented length, ft	3.28		3.05	1.55		1.55
6	Weight of core, lb	4.681		4.47	30.25		29.50
7	Weight of water displaced, lb	1.713		1.634	11.43		11.375
8	Weight of split sample, lb	2.41		2.10	15.75		14.93
9	Specific gravity	2.73		2.74	2.64		2.59
10	Diameter of core, ft	0.1075		0.1075	0.37		0.37
11	Recovery of sample, per cent. by weight	92.0		94.0	94.0		99.0
12	Copper in core, per cent	3.05		2.45	9.49		2.65
13	Recovery in pull, per cent		93.0			95.0	
14	Copper-core combined, per cent		2.76			6.11	
15	Outside diameter of hole, ft		0.167			0.49	
16	Weight of sludge, lb		14.261			57.20[a]	
17	Recovery of sludge, per cent. by weight			98.0		98.0	
18	Copper in sludge, per cent		2.70			6.05[a]	
19	Weight of core and sludge, lb		23.412			116.95	
20	Total over-all recovery, core and sludge, per cent. by weight		99.0			99.5	
21	Copper in core and sludge, per cent		2.72			6.08[a]	
22	Iron in core, per cent				9.40		3.73
23	Iron in total core, per cent					6.59	
24	Iron in total sludge, per cent					10.94	
25	Actual weight of sludge, lb					60.90	
26	Copper in total sludge, per cent					5.78	

[a] Corrected.

The diamond-drill pull from 697.62 to 703.95 ft. will be taken first, and the method of obtaining the figures in the table will be explained where necessary:

3. Pull length = 703.95 − 697.62 = 6.33 ft.

11. Percentage recovery of sample =

$$\frac{\text{Weight of core} \times 100}{\pi\left(\frac{\text{dia. of core}}{2}\right)^2 \times \text{sp. gr.} \times \text{length represented} \times 62.5}$$

(Density of sludge is assumed to be the same as that of the recovered core throughout calculations.)

13. Percentage recovery of pull =

$$\frac{\text{Sum of sample weights} \times 100}{\pi\left(\frac{\text{dia. of core}}{2}\right)^2 \times \text{sp. gr. by length of pull} \times 62.5}$$

14. Percentage of copper in core combined =

$$\frac{\text{Weight of sample 243} \times \text{per cent. of copper} + \text{weight of sample 224} \times \text{per cent. of copper}}{\text{Total weight of core in pull}}$$

15. This figure is very difficult to determine. In a diamond-drill hole the clearance between the crown and the core probably equals the clearance between the crown and the outside of the hole. With a shot drill, however, the speed at which the water is being pumped down the hole probably affects the diameter of the hole, inasmuch as variations in water speed cause crowding and thinning of the shot on the outside of the bit. The effect of crowding shot to the outside of a bit probably varies directly with the hardness of the ground, but it can well be imagined that in a soft mica schist, in which some of the richest ore occurs, it would be possible to get two or three times more outside clearance than inside clearance by crowding the shot to the outside of the bit.

17. Percentage recovery of sludge:

$$\frac{\text{Weight of sludge} \times 100}{\left\{\pi\left(\frac{\text{dia. of hole}}{2}\right)^2 \times \text{sp. gr.} \times \text{length of pull} \times 62.5\right\} - \text{weight of core}}$$

(See remarks under 15.)

20. Percentage recovery of core and sludge =

$$\frac{\text{Weight of core} + \text{weight of sludge} \times 100}{\pi\left(\frac{\text{dia. of hole}}{2}\right)^2 \times \text{sp. gr.} \times \text{length of pull} \times 62.5}$$

21. Percentage of copper in core and sludge =

$$\frac{\text{Weight of total core} \times \text{per cent. of copper in combined core} + \text{weight of sludge} \times \text{per cent. of copper in sludge}}{\text{Weight of core} + \text{weight of sludge}}$$

$$= \text{percentage of copper for the 6.33 ft. of pull.}$$

In the shot-drill pull from 596.55 to 599.65 ft., the procedure is the same as in the diamond-drill calculation, with the exception of the fact that the sludge weight and the sludge assay must be corrected for the iron introduced into it in the shape of shot.

Correcting Shot-drill Sludge for Introduced Iron

16. Corrected weight of sludge E is obtained as follows:

Where A = weight of -30 sludge plus free iron = ... 60.99
B = weight of sample cut from A = ... 3.729
C = weight of -60 sludge minus the weight of iron removed by magnet = ... 3.663
a = per cent. copper in sludge = ... 5.78
b = per cent. iron in sludge = ... 10.94
c = per cent. iron in core = ... 6.59

Substituting in the formula:

$$E = \frac{AC(100 - b)}{B(100 - c)}$$

$$E = \frac{60.99 \times 3.663 \times (100 - 10.94)}{3.729 \times (100 - 6.59)}$$

$$= 57.2 \text{ lb.}$$

The corrected percentage of copper in sludge e is obtained by substituting in the formula:

18. $$e = \frac{ACa}{EB}$$

$$= \frac{60.99 \times 3.663 \times 5.78}{57.2 \times 3.729}$$

$$= 6.05 \text{ per cent. copper.}$$

The figure of 6.08 per cent. copper is the copper content for the 3.1 ft. of hole between 596.55 and 599.65, and is obtained in the same way as for the diamond drill; see note on 21 in diamond-drill calculations.

Calculations for Grade of Orebody

In calculating an average value for a series of pulls a number of formulas are used and their merits are discussed below:

(*a*) By $\frac{\Sigma(\text{Lengths represented} \times \text{Cu per cent.})}{\Sigma \text{ Lengths represented}}$ . . . (core)

(*a*1) By $\frac{\Sigma(\text{Lengths recovered} \times \text{Cu per cent.})}{\Sigma \text{ Lengths recovered}}$ . . . (core)

(*b*) By $\frac{\Sigma(\text{Actual weights} \times \text{Cu per cent.})}{\Sigma \text{ Actual weights}}$ . . . (core)

(*b*) By The same . . . (sludge)

(*b*) By The same . . . (core and sludge)

(*c*) By $\frac{\Sigma(\text{Lengths} \times \text{specific gravities} \times \text{Cu per cent.})}{\Sigma \text{ Lengths} \times \text{specific gravities}}$ . . . (core)

(*c*) By The same . . . (core and sludge)

(*d*) By $\frac{\Sigma(\text{Lengths} \times \text{sp. gr.} \times \text{recovery per cent.} \times \text{Cu per cent.})}{\Sigma \text{ Lengths} \times \text{sp. gr.} \times \text{recovery per cent.}}$ . . . (core)

(*d*) By The same . . . (core and sludge)

(Lengths used are drilled lengths or lengths represented, unless otherwise designated.)

Method a

Incorrect: Because even if core recovery is complete it does not take varying specific gravities into account. The specific gravity is important, of course, only where either the gangue varies considerably in specific gravity over an orebody or where the valuable mineral is present in sufficiently variable quantities to make the specific gravity of the ore vary over an orebody.

Method a1

Incorrect: For the same reason as in method *a*. Also, even though only actual lengths recovered and examined are used in obtaining average copper content, the sum of those lengths does not equal the drilled width of ore. When the losses are of the lower grade portion, the average content is too high, and when the losses are high grade the average content is too low. However, if there is no sludge, and the specific gravity of core does not vary much, results may be calculated by this method advantageously as long as the percentage recovery of core is also reported.

Method b

Incorrect: Because although varying specific gravities are taken into account, varying diameters are not. Even if there is complete recovery the result may be unreliable, as undue weight may be given to a pull where the hole varies in diameter, but in a formation in which the hole does not vary in diameter the formula is sound.

Method c

Correct: Unless recovery varies but emphatically incorrect when water return is incomplete. Sludge should not be used in calculation of copper content unless water return is perfect or very close to perfect.

Method d

Remarks under method *c* apply to this method also. As pointed out earlier, the percentage of sludge recovery is a figure that is difficult to rely on, as the diameter of hole may vary and give one an apparent excess of sludge or apparent 100 per cent. sludge recovery when water return is incomplete.

Briefly, it is safer to take core results by themselves than to try to combine core with unreliable sludge, and the reliability of the sludge depends more on complete water return than on the percentage of sludge recovery figured from the theoretical size of the hole.

Churn-drill Sampling

In some of the earlier work on the concession, churn drills were used for sampling the ore horizon, and from the point of view of reliability

the samples taken by them were good. Unfortunately, however, the beds were very much folded and drilling through the ore horizon with churn drills was stopped, as it was essential to have core from which

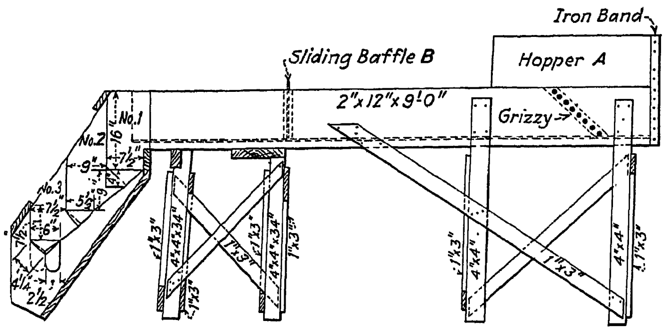

FIG. 7.—SAMPLE SPLITTER.

to interpret true widths of ore. A great deal of experimental work on sampling was done with these drills before their use as sampling machines was abandoned.

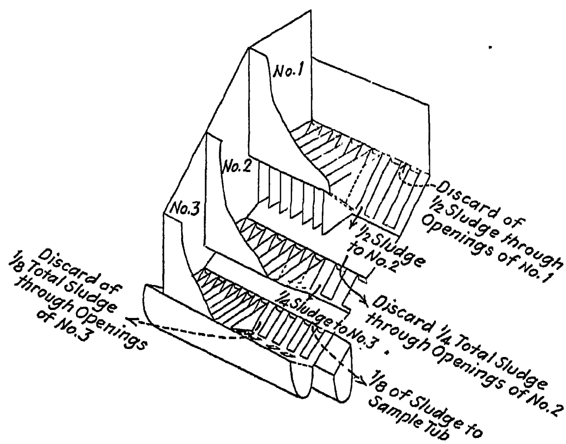

FIG. 8.—DIVIDER IN SAMPLE SPLITTER.
Right side and a few partitions removed.

The ore horizon in which the machines were used was a calcareous shale, more or less schistose, which gave very fine slime, and the principal difficulty was to arrive at some drilling procedure that would not be too slow and that would still give time for the slime to settle in the hole, so that it could be bailed out. Water level in individual holes was usually constant, and varied in different holes from 10 to 70 ft. from surface.

On encountering ore, the hole is cased to the bottom and thoroughly cleaned, and the measuring line is run before drilling is continued. It is

the practice to keep the casing not more than 30 ft. behind the tools, which necessitates either lowering a string of smaller casing or underreaming the first string of casing down after drilling 30 ft. of the ore horizon. This practice was adopted because the top of the string of tools caused caving if allowed to swing against the side of the hole.

Figs. 7 and 8 show the type of sample splitter used, which is based on an idea of Henry Krumb's worked out by R. C. Nowland on the property of the Ray Consolidated Copper Co. in Arizona. The bailer dumps into the hopper *A* while the baffle plate *B* is kept shut. As soon as all the sludge is in the hopper, the baffle is raised about 3/8 in. and the sludge allowed to run at a steady rate into the cutters. The cutters are of the Jones type, in three tiers, with launders so arranged that either 50 per cent., 25 per cent., 12.5 per cent. or all the sludge can be caught.

Drilling Procedure

After many experiments with different types of bailers of both the suction and regular dart-valve types, and with different numbers of bailings, the following procedure was found to be most satisfactory.

The casing having been set at the top of the ore horizon and the hole cleaned and measured, the tools were lowered into the hole and a distance of between 2½ and 3 ft. was drilled. The hole was then cleaned by taking 10 or more bailings with the suction bailer. This sludge was caught and treated as one sample. The tools were then lowered, an equal distance was drilled and the hole was again bailed. This process was continued as long as daylight lasted. More than four samples have seldom been taken in one day.

On the completion of the last sample for the day, about 4 lb. of lime were put into the hole in a paper bag and pushed to the bottom with the tools or bailer. The hole was allowed to stand all night and in the morning the bailer was run for two or three sets of five bailings each, with ½-hr. wait between each set of bailings. This clean-up sample was caught separately. Care was taken to hoist the bailer slowly for the first 20 ft., when taking the clean-up sample, so as to avoid agitating the fine slime unduly.

In calculating the value of the section drilled during each day, the sum of the weights of the first samples multiplied by their assay values, and the weight of the clean-up sample multiplied by its assay value, is divided by the total weight of the samples.

Sample Drying

The samples were dried in metal tubs over an open fire. This method proved satisfactory as long as the sludge was stirred continually, so that it did not cake on the bottom of the tubs and allow the sulfides to oxidize.

Reliability of Samples

In estimating the reliability of the samples taken, it is difficult to decide on a diameter for the hole, as a churn drill will drill holes of slightly different sizes in formations of different hardnesses. For this reason it is probably better to rely on close observation of the bailings from which to tell when recovery of sludge is complete, rather than to distrust a sample because it appears to be over or below the theoretical weight, unless there is some contributary cause for distrusting it.

Since all sludge from a core-drill hole is thoroughly classified and that from a churn-drill hole is thoroughly mixed, the losses in the two methods must not be treated in the same way. Even a large loss from a churn drill might have no effect on the average copper content of recovered sludge, while even a small loss from a core drill might give an entirely wrong value.

Conclusion

The writers strongly advise any engineer who is starting on a drilling campaign that demands the type of sampling described herein to use the metric system throughout. It is to be hoped that some day any other system will be illegal, but until that time arrives it is far more simple to convert to feet and tons at the end of a calculation than to use feet, inches, tons, pounds and ounces all the way through it. It will be difficult to change to the metric system after any other system is well established.

Acknowledgment

The writers acknowledge their indebtedness to the management and staffs of the Roan Antelope Copper Mines, Ltd., the Rhodesian Selection Trust, Ltd. and the Mufulira Copper Mines, Ltd., for allowing them facilities for research and help in the collection of information. Too many men have given advice and assistance for individual mention, but the writers hope that this acknowledgment will be thought adequate by those who helped in this work.

Finally, no paper on sampling in these fields would be complete without an expression of the writers' indebtedness to Mr. R. M. Geppert, of the Selection Trust, Ltd., who was the pioneer of efficient drill sampling here.

DISCUSSION

(Scott Turner presiding)

L. D. Cooper, Minneapolis, Minn. (written discussion).—This paper is an exceedingly interesting description of drill-hole sampling, and the methods in the collection of core-drill sludge are a distinct contribution to the technology of sampling. Of particular value to the engineer or geologist charged with the responsibility of making accurate estimates of ore shown up by drill holes is the equipment used and the attention given to the amount, velocity and proper return of the circulating water. The

arrangement of sludge boxes results in greater efficiency in the collection of sludge samples and, while the installation is not suitable for most underground drilling, and would be rather expensive for shallow holes, its value under suitable conditions has been proved.

Matson and Wallis have advocated a method of combining or averaging the separate analyses of core and sludge samples, which seeks to reduce the sources of possible errors in the volumetric method. In using the weight of the core and sludge recovered instead of their volumes, they have eliminated errors resulting from inaccurate measurements of the core and from variations in the diameters of both core and hole. They have introduced a possible source of error when they assume that the sludge sample is reliable only in proportion to the percentage recovered. They themselves state that "the reliability of the sludge depends more on complete water return than on the percentage of sludge recovery figured from the theoretical size of the hole." Would not the authors be more consistent if they applied a correction factor equal to the percentage of water returned? It would seem to be safer practice to assume that the analysis of the sludge recovered is representative of all the material recovered which is not core than to base its reliability entirely on the weight recovered.

A number of years ago, at an iron-ore property in Michigan, a series of tests was made of the efficiency of a new type of sludge box. For the average of the tests made, the box collected 84 per cent by weight of the sludge that was washed out of the hole, while the analysis of this 84 per cent was within 0.012 per cent of the analysis of the entire sludge. Had the analyses of core and sludge been combined on the basis of weights, a serious error would have been introduced, because there was considerable difference between the core and sludge analyses. This example is cited not to justify the loss of 16 per cent of the sludge sample but to show that methods suitable for one operation may not be applicable to another.

Inasmuch as the principal purpose of most core-drill work is to locate orebodies and to determine their tonnages and mineral contents, sampling and the interpretation of drill-sampling results should be considered in relation to ore estimates. Matson and Wallis, in their formula *d* on page 76, have introduced factors of specific gravity and percentage of sample recovery in determining grades from various analyses of a single drill hole or of a number of holes. Their use of percentage of recovery in determining average grades is open to question. The percentage of recovery is not only based on the theoretical size of the hole but on the expectation that the specific gravity of the sludge will be the same as that of the core—too great an assumption in many orebodies. However, we need not be too concerned about this, for, if in formula *d* we substitute for "recovery per cent" the formula under 20, page 75, formula *d* is reduced to

$$\frac{\sum \dfrac{(\text{Weight of core} + \text{weight of sludge}) \times \text{Cu per cent}}{\text{Area of hole}}}{\sum \dfrac{\text{Weight of core} + \text{weight of sludge}}{\text{Area of hole}}}$$

so that the average grades are determined on the basis of the weight of the sample recovered in relation to the area of the hole from which the sample came. The factors of specific gravity and percentage of sample recovery have cancelled out and have no influence on the final result.

Mr. Joralemon[2] has shown that at Ajo, Ariz., the standard practice of combining assays of core and cuttings on a volume basis produces results that are sufficiently accurate for all commercial purposes. Channel samples of test pits checked the diamond-drill sampling within 0.005 per cent in carbonate ore, averaging 1.54 per

[2] I. B. Joralemon: The Ajo Copper Mining District. *Trans.* A. I. M. E. (1914) **49**, 605–606.

cent copper and within 0.05 per cent in sulfide ore, averaging 1.50 per cent copper. The core recovery in this drilling was only 30.3 per cent. Millions of tons of ore have since been mined from this orebody and original estimates have been substantiated.

I. B. JORALEMON, San Francisco, Calif. (written discussion).—The method of interpreting drill samples described by Messrs. Matson and Wallis is a remarkably careful and painstaking attempt at accuracy. Unfortunately it seems very doubtful whether the sample itself is nearly as accurate as the calculations made from it.

There can be no question that drill holes have a greater diameter at places where the rock is hard than where the rock is soft. If the variations occurred over even sample lengths, the Rhodesian method would go far toward compensating for them, but the most serious variations come where there are narrow soft seams or bands in the rock that occupy only part of a sample interval. In such cases there is often no core at all. The amount of sludge from the soft bands is much greater than is indicated by the diameter of the hole estimated from adjoining lengths where core is recovered. Therefore when core and sludge assays are combined in proportion to their weights, undue importance is given to sludge from bands where there is little or no core. In most orebodies this would result in an accepted grade that is too high.

Where the recovery of sludge is not complete, and sludge assays are discarded entirely, too much emphasis is given to the harder parts of the orebody, which core well. It seems safer to give at least some weight to the sludge assay where core is incomplete, even if the return of sludge is also imperfect.

There is no possible way of obtaining a result that is theoretically perfect unless the diameter of hole and of core could be measured and a separate sample taken every time either of these diameters varies. This, of course, is out of the question; hence any method of calculating drill-hole results is an approximation. The problem is to select a method that is as accurate as the sample itself, and that does not, by its laboriousness or intricacy, give the impression of being more accurate than the basic data permit.

The method described by R. D. Longyear,[3] and used at Ajo and elsewhere, seems to meet these requirements much better than the Rhodesian method. Diameters of core and hole are assumed to be constant, and core and sludge assays are easily combined by a table or chart showing proportionate weight to be given each sample for varying lengths of core recovered. This method compensates, at least roughly, for excessive sludge recovery caused by caving from soft bands, and for small sludge recovery due to loss of part of the water in fissures. Obviously the method is not perfect, but it avoids the dangerous overemphasis on sludge samples of the Rhodesian method. And it saves a great amount of detailed figuring that means little or nothing because of the inadequate grounds on which it rests.

Far more misleading than any possible error in combining samples in one drill hole is the assumption that a comparatively small number of holes, spaced 400 to 1000 ft. or more apart and varying from 2.5 to 6 or even 15 per cent copper, represent the grade of ore that will be mined to within 0.01 per cent. It is a well-known principle of all sampling that in order to give a fair assay value of an orebody that varies considerably in grade, there must be a great number of closely spaced samples. In all save perhaps the most thoroughly developed portion of the Roan Antelope orebody, drill holes in Rhodesia are so far apart, and the grade of ore is so irregular, that this requirement is not met. Drilling has shown that there are extremely valuable orebodies, but the actual grade of ore may vary from the estimated grade by several tenths of one per cent. The meticulous method of calculating drill results is only too likely to divert attention from this uncertainty.

[3] R. D. Longyear: Diamond-drill Sampling Methods. *Trans.* A. I. M. E. (1923) 68, 423.

Ventilation at the Portovelo Mines, Ecuador

By John P. Harmon,* Tucson, Ariz.

(New York Meeting, February, 1931)

This paper was written with two objects in view: (1) To describe in detail what has been done toward the ventilation of the main unit of the Portovelo mines and the results; (2) to give information that may be useful to anyone who may have to make a preliminary mine ventilation survey, with recommendations for the improvement of the existing system and finally the maintenance of the improved system.

General Description

The mining operations of the South American Development Co. at Portovelo, Province of El Oro, Ecuador, are described here only briefly, as a full discussion does not fall within the scope of this paper. The orebodies are of the steeply dipping, gold quartz vein type, averaging about 2.0 meters wide. As shown in Fig. 1, the ore chutes are relatively far apart, making considerable development work necessary before connecting raises can be driven between the levels. Faulting[1] on a large scale and a parallel vein system have made long dead-end drifts and many crosscuts necessary in the exploration of veins. The ventilation of these dead ends is described under Auxiliary Ventilation. The method of mining[2] is the filled rill stope method, similar to the method that is largely employed at the Butte mines.

The main shaft of the mine is in the Amarillo Valley, on either side of which mountains rise abruptly to a height of several hundred feet above the collar of the shaft. Entrance is made to the mine: (1) Through the American shaft, which extends vertically from the surface to a depth of 1100 ft., of which only the first 700 ft. are in active use at the present time; (2) levels are driven from the American shaft at intervals of 30 m. (100 ft.), of which only the third, fifth and seventh levels need be considered in the ventilation problem; (3) adits at intervals of 30 m. (100 ft.), with a few exceptions, above the collar of the American shaft.

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* Formerly Mining Engineer with the South American Development Co., Portovelo, Ecuador.

[1] P. Billingsley: Geology of the Zaruma Gold District of Ecuador. *Trans.* A. I. M. E. (1926) **74**, 255.

[2] R. Emmel: Mining Methods in Zaruma District, Ecuador. *Trans.* A. I. M. E. (1925) **72**, 447.

The main adits on the south side are $\frac{3}{4}$ level adit, A level adit and B level adit. Those on the north side are A level adit, E level adit, E$\frac{1}{4}$ level adit and F level adit.

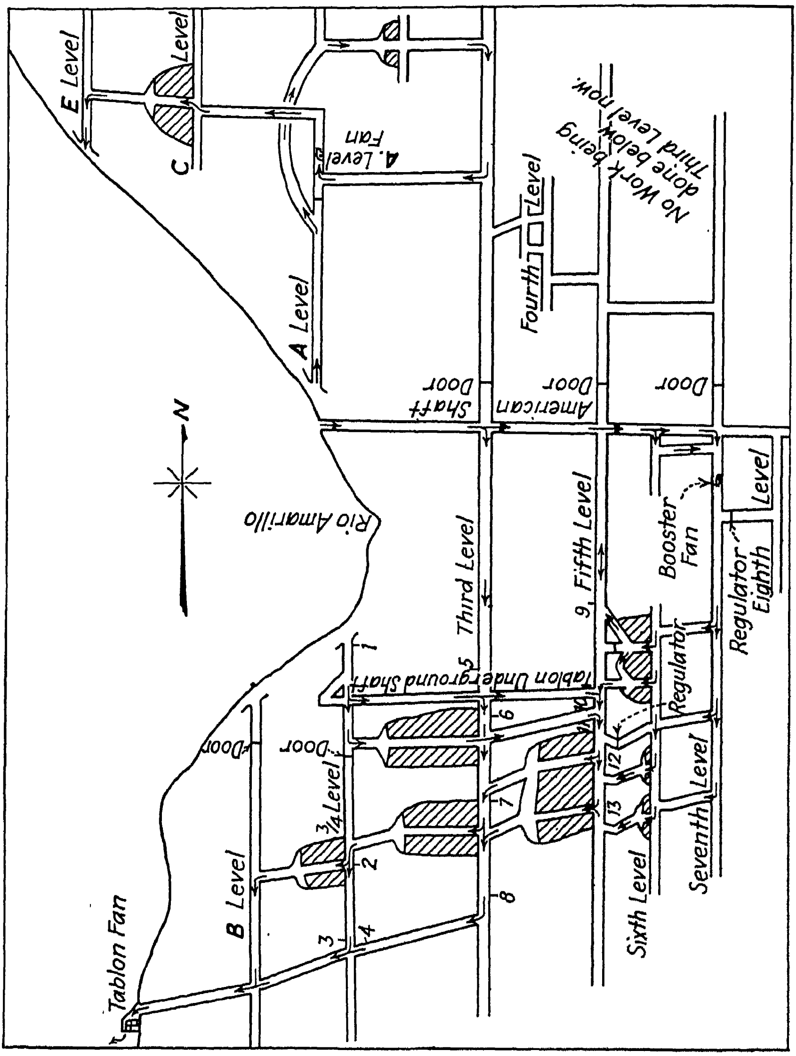


FIG. 1.—PROFILE OF PORTOVELO MINE WORKINGS CONSIDERED IN VENTILATION PROBLEM.

SURFACE TEMPERATURE

A record of the maximum and minimum daily temperature readings has been kept for a period of 30 years. This record shows that the average maximum temperature for the last four years has been 80.8° F. and the average minimum 68.2° F. It also shows that the difference between maximum and minimum monthly average for the last four

years has ranged between 6° and 20° F. Table 1 shows the maximum and minimum average monthly temperature for the years 1927, 1928, 1929 and 1930 to date. The temperature readings were taken in the shade and indicate only the air temperature, which is not high enough to require refrigeration for mine ventilation purposes, nor low enough to materially change the mine temperature.

TABLE 1.—*Monthly Average Maximum and Minimum Temperature as Recorded at Portovelo, Ecuador*

Year		Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.
1927	Max...	75.8	76.4	78.0	78.9	80.4	80.9	82.5	81.5	80.1	81.5	81.9	82.6
	Min...	68.8	69.5	69.8	69.5	69.3	67.6	65.3	66.5	67.6	68.3	68.9	68.9
1928	Max...	77.0	77.0	78.5	79.0	78.5	78.5	81.1	80.6	82.5	83.5	82.6	81.4
	Min...	67.9	68.1	67.6	68.0	68.5	66.7	64.2	65.6	67.1	67.9	67.7	68.3
1929	Max...	79.5	77.2	78.6	79.6	78.8	78.6	81.5	81.2	82.9	82.3	81.6	80.2
	Min...	67.2	67.9	68.0	67.5	67.7	66.3	64.1	65.7	67.1	67.2	67.5	68.0
1930	Max...	79.5	77.4	79.0	79.2								
	Min...	67.8	67.4	68.0	67.8								

HUMIDITY

At the latitude of the mine there are two distinct seasons of the year; the wet season, December 15 to June 15, and the dry season, June 15 to December 15. During the wet season rains occur daily, averaging about 60 in. of rainfall per year. In the dry season there is very little precipitation. (See Table 2.)

The humidity of the air on the surface is higher than might be expected, especially during the dry season. A record of the relative humidity at the collar of the American shaft shows that the average for the period July 15 to August 15 was 48 per cent.; that from May 8 to June 8, 64 per cent.

The humidity of the mine air, however, is independent of the surface humidity and the air attains a high relative humidity soon after entering the mine. The high humidity is due to:

1. Mine water. Enough water seeps from the rocks to keep the back of many of the drifts and stopes permanently wet, and in some places there is such an excess as to form drippers.

2. Underground streams. In several places underground streams have been tapped, which have tended to saturate the air.

3. Water from drills. To reduce the possibility of the miners contracting silicosis, dry drilling is prohibited. Although there is less water

from wet drilling than from the sources mentioned above, it tends to keep the relative humidity at the working places very near 100 per cent.

4. Perspiration from men and animals underground. This source of moisture, although small, locally tends to saturate air of low velocity that might otherwise be fairly dry.

TABLE 2.—*Portovelo Rainfall Record, in Inches*

Year	Jan.	Feb.	Mar.	Apr.	May	June	July	Aug.	Sept.	Oct.	Nov.	Dec.	Total
1927	17.45	12.35	10.95	3.25	1.50	0.35	0.0	0.05	2.05	2.10	1.30	2.10	53.45
1928	10.85	9.65	23.10	22.75	8.10	2.10	1.10	0.0	0.10	0.20	1.50	4.45	83.90
1929	7.65	14.55	11.00	8.70	4.98	1.35	0.0	0.0	0.85	1.35	2.70	3.25	56.38
1930	6.25	13.45	17.05	13.40									50.15

VENTILATION IN THE PAST

The system of ventilation in operation before the changes described in this paper were made was as follows:

A Sturtevant, Silentvane No. 95, design 2, double-width double-inlet fan, located at ¾ level portal, forced fresh air along the drift (Fig. 1). Some air passed up through the ¾ level stopes and out to the surface; the remainder passed down to third level through two operating stopes. From third level the air was forced to fifth level through two operating stopes, and then to seventh level through a manway. The air on third level and fifth level was kept from returning to the American shaft by doors in the drift south of the shaft. On seventh level the air went north, past the American shaft, where it was diluted with fresh air.

A booster fan of local make was placed on seventh level, north of the American shaft, and a Sturtevant fan similar in design and capacity to the one at ¾ level portal was located at F level portal, in the north end of the mine (Fig. 7). These two fans working in series exhausted the air from the mine. The American shaft was kept downcast, to preserve the shaft lining, which at that time was wood.

Under this system the maximum amount of air circulating in the mine was 33,000 cu. ft. per minute, as all of the fans operated in series.

Air splits are considered essential in good ventilation practice, and in some parts of the United States they are required. As stated later, the new system of ventilation was based on this principle, while the old system worked on a series plan, using the same vitiated air over and over again.

PRELIMINARY CONCLUSIONS

After a preliminary study of the problem of ventilating this mine the following conclusions were reached:

1. That the pressure system employed in the south part of the mine should be changed to an exhaust system. This change was decided upon because: (*a*) It was desired that air splits be employed as frequently as practicable, in order that partly vitiated air might be diluted before passing to the next working place. The use of air splits, while possible with a pressure system, can be handled more easily with an exhaust system. As was noted under Ventilation in the Past, the same air was used over and over, so that it was thoroughly vitiated on reaching the exhaust passage. (*b*) With the two fans in series, as in the original set-up, approximately 33,000 cu. ft. of air per minute was circulated through the

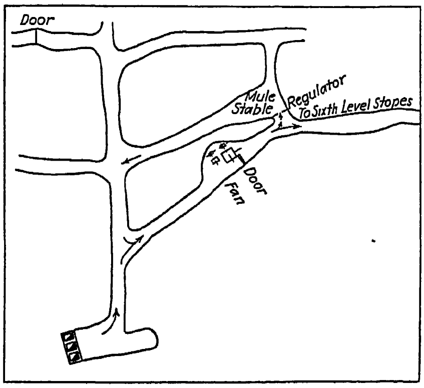


FIG. 2.—PLAN OF SEVENTH LEVEL STATION AND FAN SITE.

mine, while with the same two fans exhausting independently twice as much air was circulated.

2. That a line of raises should be driven in the south part of the mine, to be used primarily for ventilation purposes.

3. That the fan which had been forcing air into the south part of the mine should be moved to the Tablon shaft, where the air raise from B level to the surface had broken through.

4. That the booster fan formerly mounted north of the American shaft on seventh level should be moved south of the shaft, to force air south to sixth level stopes, as well as deliver some air north through the mule stable (Fig. 2).

5. That a system of air splits should be adopted as far as possible.

6. That solid platforms in raises to be used for airways should be replaced by lattice platforms.

7. That it would not be practicable to reduce the humidity of the mine air.

8. That, due to the high humidity and prevailing rock temperatures, large quantities of air at high velocity would be necessary at all working places.

9. That it would not be advisable to precool the air to be used for ventilation.

Fan Installations

An untimbered inclined raise was driven from B level to the surface, about 700 m. (2300 ft.) south of B level portal. A Sirocco fan designed to exhaust 60,000 cu. ft. of air per minute with 3½-in. water gage has been installed in the fan house at the collar of this raise, replacing the 33,000 cu. ft. per minute fan that was placed there as a temporary installation. The fan house was constructed of crosslapped boards, gunnited on the outside to make the walls fire-resisting as well as airtight. The roof

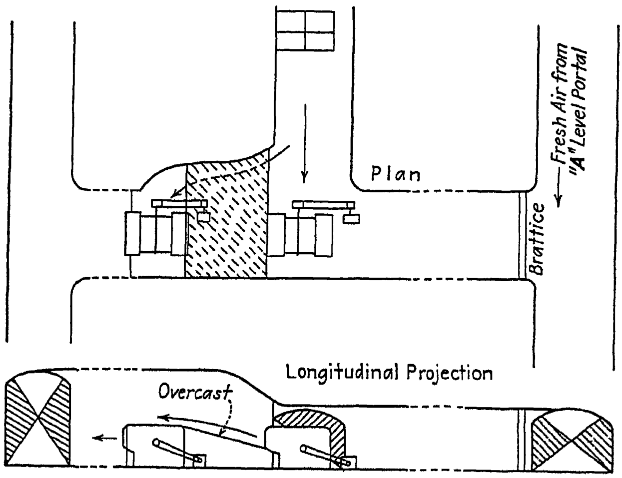


Fig. 3.—Parallel set-up of two exhaust fans on A level.

was made airtight by inserting a layer of tar paper between the crosslapped boards.

Two Sturtevant fans, each designed to deliver 33,000 cu. ft. of air per minute against 3½-in water gage, are to be installed in the north part of the mine. Fig. 3 shows that these fans can be operated either in series, in parallel, or singly, depending upon conditions in the mine as they change from time to time. Changing conditions will determine the mode of running fans. Tests were made, and at the present time the fans will operate best in parallel. Later, as resistance in the ventilation circuit increases, it may be necessary to run the fans in series. In general, if the mine has high resistance, two fans will deliver more air when operating in series; with low resistance, two fans will deliver more air when operating in parallel.[3]

[3] W. S. Weeks: Ventilation of Mines, 122. New York, 1926. McGraw-Hill Book Co., Inc.

At present no mining is being done in the Cantabria stopes, so that these raises and stopes are kept open for the exhaust air from the A level fans. Later, however, after the Soroche and Tamayo stopes are exhausted, the Cantabria stopes will again be worked. When men are working in the Cantabria stopes, they can be furnished with fresh air from the same fan set-up, merely by replacing the brattice shown in Figs. 3 and 7 by a regulator. The air then will enter the portal at A level, pass through the fan, up through Cantabria and out. The amount of air left circulating through Soroche and Tamayo stopes will be controlled by the regulator.

Evasé Chimney

When a fan is working as an exhauster, the air leaves the fan at high velocity, causing a high percentage loss of kinetic energy. If the air leaving the fan can be slowed down (made to rise to atmospheric pressure gradually) some of the velocity pressure is converted to static pressure.

Table 3.—*Velocity Pressure in Relation to Length of Chimney*

Length of Chimney, Ft.	Velocity Pressure Recovered, Lb. per Sq. Ft.	Length of Chimney, Ft.	Velocity Pressure Recovered, Lb. per Sq. Ft.
0	0.000	8	1.382
1	0.323	9	1.449
2	0.580	10	1.506
3	0.787	11	1.555
4	0.955	12	1.597
5	1.092	13	1.633
6	1.206	14	1.664
7	1.302	15	1.691

The Evasé chimney is an old device for recovering static pressure on exhaust fans, but I believe a description of how we designed the chimney for our fan may well be given here. The chimney should expand gradually, otherwise eddy currents are set up and the recovery is poor. Let the area at the fan discharge be A_1 and the area at the top of the chimney A_2. Let Q be the quantity flowing in cubic feet per second. Then the velocity at the throat is Q/A_1 and the velocity at the top is Q/A_2. The velocity pressures at these two points are $WQ^2/2GA_1$ and $WQ^2/2GA_2$, where W is the weight of 1 cu. ft. of air. If conversion is perfect, the velocity pressure recovered is $\frac{WQ^2}{2G}\left(\frac{1}{A_1^2} - \frac{1}{A_2^2}\right)$. As the conversion is never perfect, a factor representing the efficiency of conversion must be used. A chimney with a 10 per cent. slope was decided upon. W. H. Carrier[4] gives 70 per cent. for the efficiency of a conical diverging nozzle with a 10 per cent. slope.

[4] W. H. Carrier: Fan Engineering, Ed. 1. Buffalo, 1914. Buffalo Forge Co.

The tabulation in Table 3 was made from the above formula. These data were plotted on rectangular coordinates, from which it was apparent that an appreciable saving is made by using a chimney 10 ft. long, while beyond that point the velocity pressure recovered is not appreciable per unit length of chimney. Therefore a chimney 10 ft. long was decided upon as economical.

Auxiliary Ventilation

Here, as at other mines, the problem of furnishing fresh cool air to dead-end workings can best be solved by using small electrically driven fans, delivering air to the face through canvas tubes. At present there are two Coppus TM-6 fans and two Sirocco $2\frac{1}{2}$ Troy fans, which are used where relatively long lengths of canvas tube are needed.

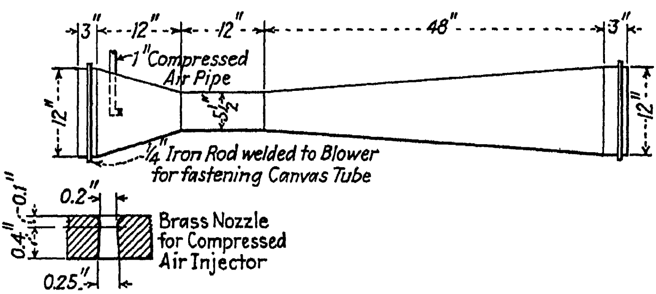


Fig. 4.—Modder blower.

Where conditions make the installation of an electric blower impracticable, an injector blower of the Modder deep type (Fig. 4) has been found convenient and effective, when the distance for delivering the air is not over 300 ft. As these blowers, made in the machine shop at a low cost, have no moving parts, there is no maintenance cost, and since they require only a common air hose connection they can be installed quickly and easily. Where the face to be supplied with fresh air by an electric blower is too far for efficient operation, an injector blower has been found effective, installed directly in the line about one-third of the distance from the fan to the face.

The outlet of the canvas tube should be as close to the workers as convenient, never farther away than 25 ft. Hence, two short sections of tube (25 ft. and 50 ft. long) are furnished to the men in each dead end. When a face has been advanced 25 ft. beyond the last 100-ft. section of tube, a 25-ft. section is put on. After another 25-ft. advance, the 25-ft. section is replaced by the 50-ft. section, and finally the 25-ft. section is added, making a total of 75 ft. Thus the end of the tube is kept within 25 ft. of the face at all times.

When canvas tubing was first used for ventilating dead ends in this mine, the miners did not understand the necessity for keeping the tube hung properly, in good repair and free from holes, and for keeping it close to the face. The tube was placed near the working place, in spite of the workers, and now they seem to realize the improved working conditions and maintain the tube themselves.

Some time ago a considerable quantity of 12-in. Ventube was stocked. When the next order is made, it will probably be for 16-in. tube, because

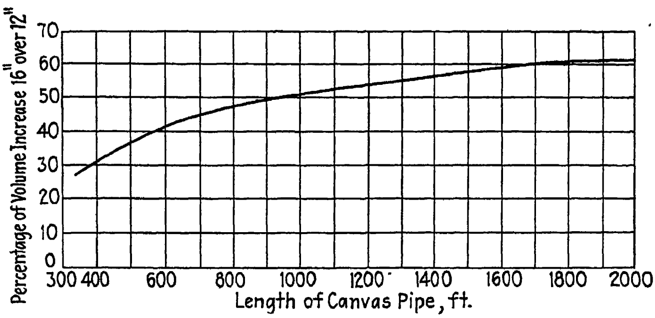


Fig. 5.—Curve showing advantage of using the larger size of canvas pipe.

of its increased efficiency (Fig. 5). For furnishing air to men working in raises while no drilling is being done, a compressed-air hose is used, with a special nozzle. The nozzle (Fig. 6) was made in the mine machine shop. The body is packed with waste to prevent free escape of air.

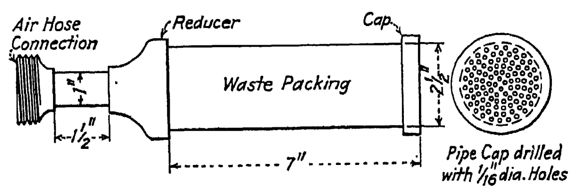


Fig. 6.—Nozzle for diffusing compressed air in raises.

As expanding air absorbs considerable heat, a given quantity of compressed air will have a greater cooling and drying effect than a like quantity of air forced to the back through canvas tubing.

Ore Transfer Raise Used as Airway

On several occasions it has been necessary to use a one-compartment long raise for an ore transfer as well as for an airway. An existing raise can be made to serve both purposes at a cost much lower than that of driving a completely new raise. This is done by driving a new small raise 15 ft. from the old one, the height depending upon the size of the old raise and the desired capacity of the ore chute. The two are then connected by a raise inclined up from the old raise, so that muck dropped

through the transfer cannot fall into and block the ventilation raise. This plan, tried twice, works well so long as the chute is kept drawn down below the connection (Fig. 7).

Because of the large quantity of water in all parts of the mine, there is no dust underground, therefore there is little objection to a high velocity in the mine airways. When the velocity is sufficient to make a candle flame flicker badly, electric lights are used.

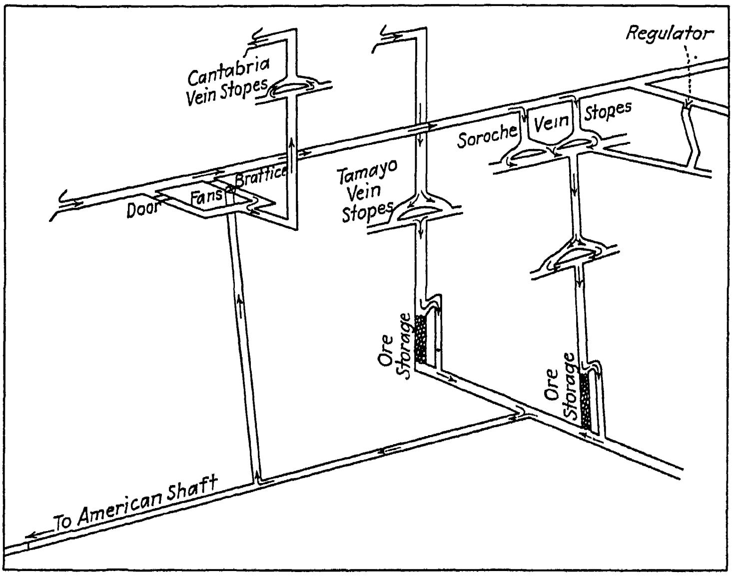


FIG. 7.—VENTILATION SYSTEM IN NORTH PART OF MINE.

AIR SPLITS

Every advantageous opening for admitting air to the mine workings is utilized. This, of course, must not be understood to mean all openings, for some entries have to be blocked by doors or brattices to force the air to traverse the lowest working level and not short-circuit. This is done for several reasons; besides putting different parts of the mine on independent air sources, the velocity in any one entry for the same total volume of air is greatly reduced, with consequent reduction in the resistance offered to the flow of air in the mine. The reduced resistance in turn allows more air to flow.

Air enters the south part of the mine through A level and 3/4 level portals, splits and goes to the lower levels through the Tablon underground shaft and an abandoned stope (Fig. 1). The former split delivers air to the third and fifth levels. while the latter delivers fresh air to the fifth level only.

The mine north of the American shaft receives fresh air through A level portal and an abandoned stope that opens into C level and D level portals (Figs. 1 and 7).

Cooling Air for Ventilation Purposes

Inasmuch as the rock temperature at the present depth of mining is not over 91° F., there is no special need for refrigeration, even locally. Sufficient fresh circulating air will cool the working places to a comfortable working temperature.

Water sprays have been tried for cooling, but were found unsatisfactory, because the temperature of the air was so nearly the same as that of the cooling water that very little cooling resulted from letting the air pass through a number of sprays.

Mine Fires

It is believed that there is little danger of serious fire in this mine, for the following reasons: (1) Little timber is needed, as the cut and fill method of stoping is used; (2) the water in the rock cavities and the high relative humidity of the mine air keep a great part of the timber damp; (3) green timber is used; (4) all timber is of hard wood; (5) there is very little timber in the hoisting shafts. The American shaft is concrete cribbed and the guides in the Tablon underground shaft are carried on stulls.

The fire hazards might be listed as follows: (1) Underground store houses; (2) electrical equipment in the mine; (3) accumulated inflammable refuse left on the levels.

The fan houses are designed so that the direction of air in the mine can be reversed if necessary. A list of contingencies that would necessitate the reversal of the fans is too long and varied to be of interest here. The decision as to whether the fan should be reversed or not should be left to some one thoroughly familiar with the air currents in the mine and the effects that the reversal would have upon the existing currents.

In making any decision as to changing the direction of air flow by manipulating the fan, the questions to be considered are: Will the new condition create a passage that will be free from smoke and gas, through which men may leave the mine? Will the passage so created differ materially from that generally used by the men as they leave the mine? This consideration is important, for the men are liable to be excited in case of fire and to choose their usual exits, regardless of signs or warnings that may be posted to the contrary.

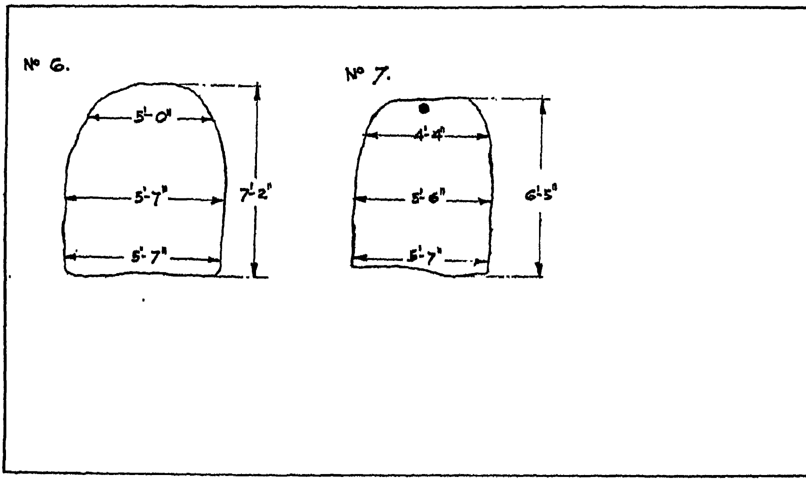
Method of Taking and Recording Observations Underground

It has been the writer's object, in computing quantities underground to get relative volumes rather than exact volumes of air. This was done

because: (1) The purpose is not to furnish a specified quantity of air to each working place, governed by the number of men working there, but to furnish sufficient excess to cool the place to a comfortable working

Working or Location	Mark.	Vel. ft/min	Area	Temperature.		Air Condi-tion	Personal Reaction.	Remarks.
				W.B.	D.B.			
¾ Portal ¾ L				74½	77	A	Too cool for comfort.	Intake Air.
	1	742 735	Same			A	ditto	Water in ditch. Walls from half way up side are dry.
	5.	450 .452.	Same	74½	77.	A	ditto.	Very dry here. No water at all.
4-28-30	2.	680. 675.	Same.	73	73	A	ditto.	Water on floor and in ditch. Walls damp.
	6.	155 162	See over	79½	80	B	Warm but not uncomfortable	Floor wet. Water in ditch. Walls Fairly dry. Timbered.
	7.	900 925	See over					

a



b

FIG. 8.—MINE RECORD SHEET.
a. Front. *b*. Back.

temperature; (2) it was desirable to make observations quickly and thus complete a ventilation survey in the shortest possible time.

Readings are taken at the same place in each measured cross-section, for periods of one minute, until two readings check reasonably well.

Whenever the observer takes a reading at one of these stations he assumes a standard position; namely, facing at right angles to the direction of air flow, with the anemometer held at arm's length in the center of the drift and about one-seventh of the height below the back. Although this does not give a true value for the quantity of air passing the particular point, it gives a relative value which will show an increase or decrease in volume, as the case may be.

Width and height are measured at desirable cross-sections in the drifts. These measurements are plotted and their area found by a planimeter. The positions are numbered, their numbers being marked on the wall of the drift or side of the raise, and are called ventilation stations; each is marked on a profile of the mine workings, the directions of the air currents being also indicated. This profile is kept up to date, showing all raises, stopes and drifts in the mine that are important with respect to ventilation.

Fig. 8 is a copy of a mine record sheet. Separate sheets are used for each set of readings.

When computing the exhausting power of the main fan more careful observations are taken. The anemometer is operated from a rod 3 ft. 6 in. long, the clutch of the instrument being manipulated by the observer by means of wires reaching from the instrument to the handle of the rod.[5]

Conclusions

After observing the ventilation in this mine, the writer feels qualified to make the following suggestions:

1. To spend a large sum of money to make a few main drifts cool will not give efficient ventilation and certainly will not justify the cost.

2. To ventilate stopes two openings must be maintained, one on the high-pressure and one on the low-pressure side, so that the air will circulate. This may seem obvious, but its importance often is not appreciated.

3. To be effective the canvas tube that delivers air from a small fan or blower to the face must be kept free from kinks, free from leaks and its end within 25 ft. from the workers that it is meant to serve.

It is regretted that no temperatures were taken in the mine before the 33,000-cu. ft. fan was moved from ¾ level portal to the present fan site on Tablon. Records were not taken because the old system was obviously so wrong that there was no doubt in the management's mind that the change would be for the better. A record was made, however, of the temperatures in the mine (April, 1929) soon after the fan was moved. Temperatures were again taken at similar positions in the mine after the 33,000-cu. ft. fan had been replaced by a 60,000-cu. ft. fan.

These comparative temperatures are given in Table 4. The location of the various ventilation stations is shown in Fig. 1. The most apparent

[5] For methods of determining exact quantities of air flowing through mine airway see G. E. McElroy: Why, When and How to Make Ventilation Surveys of Metal Mines. U. S. Bur. Mines *Circular* 6086 (1928).

drops in temperature are at No. 5, a main haulageway, where the drop was from a dry-bulb temperature of 87° to 74.5° F.; at No. 8, the main return air course from the lower levels, where the drop was from 88° to 83° F.; at No. 10, a main split of fresh air to working stopes, where the drop was from 87° to 75° F. These comparisons serve to show the change in air temperature in the mine and emphasize the necessity of air splits.

TABLE 4.—*Temperatures Recorded in the Portovelo Mines*
See Fig. 1 for Positions of Stations

Stations	April, 1929	May, 1930		Difference
	Dry-bulb Temp., Deg. F.	Wet-bulb Temp., Deg. F.	Dry-bulb Temp., Deg. F.	
1	75	74½	80½	
2	83	80½	80½	2½
3	84	82	82	2
4		82½	82½	
5	87	74½	74½	12½
6	80	75½	75½	4½
7	86	79	79	7
8	88	83	83	5
9	87	86	86	1
10	87	75	75	12
11	86	77	77	9
12	87	77	77	10
13	88	83½	83½	4½

The writer does not wish to convey the idea that these temperature drops are entirely due to changing the mine from a pressure to an exhaust system. He does, however, contend that under the circumstances an exhaust system, in conjunction with more openings in the mine and more air splits, is responsible for the improvement that is so apparent over conditions of a year ago.

ACKNOWLEDGMENTS

The writer is indebted to W. B. Phelps, General Superintendent of eht South American Development Co. at Portovelo, Ecuador, for his kind assistance and helpful criticism; to Luther Yantis, Chief Engineer of the South American Development Co. at Portovelo, Ecuador, for helpful criticisms and the use of the company's maps and drawings in the preparation of this report, and to the South American Development Co. for providing the opportunity to study the ventilation problems at the mine.

Operation of Pressure Fans in Series

By Walter S. Weeks* and Vitaly S. Grishkevich,† Berkeley, Calif.

(New York Meeting, February, 1931)

Considerable difference of opinion seems to exist as to whether the rate of air flow when a fan is placed on a given duct should be determined by the use of static pressure or total pressure characteristics. A little study will show that both methods are correct.

Assume that a fan is connected to a duct by an expansion piece as shown in Fig. 1, and that steady flow is established. Considering the atmospheric pressure as zero pressure, let the static head at the outlet of the fan be h and the velocity at this point be v_1. The total head at the outlet is then $h + \frac{v_1^2}{2g}$. The total head at this point must be equal to the total head at the end of the duct plus the losses that have occurred between the two points. Let f be the friction loss in the straight duct, s the shock loss in the diffuser, and v_2 the velocity at the end of the duct. Then

Fig. 1.—Fan on duct.

$$h + \frac{v_1^2}{2g} = \frac{v_2^2}{2g} + f + s$$

A curve showing the value of the left-hand side of the equation for different rates of flow is the total pressure characteristic of the fan, while a curve showing the value of the right-hand side is the total pressure characteristic of the duct. If these two curves are plotted on the same chart, the intersection indicates the rate of flow when steady flow is established.

We may look at this problem in another way. The static pressure at the outlet of the fan is the static pressure of the fan and it is also the static resistance of the duct from that point to the end. The friction in the straight duct is f. If the duct were the same area as the fan outlet, f would be the static resistance of the whole duct system. In the present case, however, this resistance is decreased by the change of velocity head into static head in the expansion piece. If conversion were perfect, the static pressure recovered would be

$$\frac{v_1^2}{2g} - \frac{v_2^2}{2g}$$

* Professor of Mining, University of California.

† Student, School of Mines, University of California.

Owing to the fact that some shock loss (s) occurs in the expansion piece the recovery is

$$\frac{v_1^2}{2g} - \frac{v_2^2}{2g} - s$$

The static resistance of the duct is then

$$f - \left(\frac{v_1^2}{2g} - \frac{v_2^2}{2g} - s\right) = f - \frac{v_1^2}{2g} + \frac{v_2^2}{2g} + s$$

When steady flow is established

$$h = \frac{v_2^2}{2g} + f + s - \frac{v_1^2}{2g}$$

This differs from the total pressure equation only in the fact that $\frac{v_1^2}{2g}$ has been subtracted from both sides.

A plot of h against rate of flow is the static pressure characteristic of the fan, and a plot of the right-hand side is the static pressure characteristic of the duct. The crossing of these curves will indicate the same rate of air flow as the crossing of the total pressure characteristics.

A recent paper[1] on the subject of fans in series condemns the use of static pressure curves in predicting the performance of fans. We can see no basis for this criticism, as the use of static pressure curves merely eliminates the velocity head at the junction of fan and pipe from both fan and pipe characteristics, in the case of the single fan.

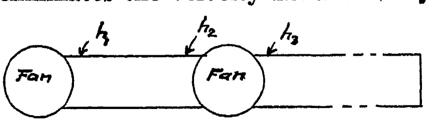


FIG. 2.—FANS IN SERIES.

Let us now consider the case of two fans operating in series, as shown in Fig. 2. Let the static head at the outlet of the rear fan be h_1, the static head at the inlet of the forward fan be h_2, and the static head at the outlet of the forward fan be h_3.

The static resistance of the connecting duct is $h_1 - h_2$. The static resistance of the duct connected to the forward fan is h_3. The total resistance of the duct system is $h_1 - h_2 + h_3$.

The static head produced by the rear fan is h_1 and the head produced by the forward fan is $h_3 - h_2$. The static pressure produced by the two fans is $h_1 + h_3 - h_2$, which is identical with the duct resistance.

If, then, we add the static pressures produced by the fans when in series, for different rates of flow, we have the combined static pressure curve of the fans, and if we add the static resistances of the two parts of the duct system, for different rates of flow, we have the static pressure characteristic of the duct system. The crossing of these two curves indicates the rate of flow that will take place.

[1] G. E. McElroy and A. S. Richardson: Experiments on Mine-Fan Performance. U. S. Bur. Mines *Tech. Paper* 447.

The static resistance of a duct is the velocity head at the end, minus the velocity head at the beginning, plus all the shock losses in the duct, plus the friction.

The question now arises, are the static pressure characteristics of the fans the same when in series as when the fans are operating alone?

Arguing from theory alone, the rear fan should have the same static pressure characteristic when operating in series as when operating alone, because in both cases it is taking the air from the still atmosphere and raising the pressure in exactly the same way. The forward fan, however, has a distinct advantage. The air is delivered at the inlet by the first fan and so the forward fan is relieved of the burden of accelerating its own air at the inlet. Bernouilli's theorem applied to the flow through this fan shows that if the forward fan is not required to accelerate its own air, it will impress on the delivered air a positive static head greater than it would impress if accelerating its own air, by the amount of the velocity head of the inlet air, it being assumed of course that the fan in the two cases is handling the same amount of air.

The forward fan possesses the additional advantage of handling air at slightly higher density. It is also possible that the change in stream lines of the air entering the forward fan and vibrations of the air may have some effect.

The experiments described in this paper were made to determine the effect on static pressure characteristics of placing two fans in series.

Apparatus

The arrangement of the apparatus with fans in series is shown in Fig. 3. The fans used were Monogram exhausters, size 0000, made by the B. F. Sturtevant Co. The wheel diameter was 6 in., the diameter of the inlet 3.2 in., the diameter of the discharge 2.75. Each fan was connected to the pipe by an expansion piece which was considered a part of the fan, and the increase in static pressure due to the recovery of velocity pressure in the expansion piece was credited to the fan.

The fans were belt driven by direct-current motors operated by current supplied by a motor-generator set with voltage control. The exact adjustment of speeds was accomplished by slide-wire rheostats on the motors. Each fan was equipped with a Weston electrical tachometer, the voltmeter of which was placed on the control table near the rheostats.

At the points where static pressure was desired pinhole tubes were inserted. These were connected to a vertical manometer containing alcohol, which could be read to 0.01 in. Velocity was measured with a Pitot tube connected to an inclined manometer with a magnification of 10 to 1.

Different resistances were obtained by inserting in the end of the pipe circular wooden blocks containing orifices of different sizes.

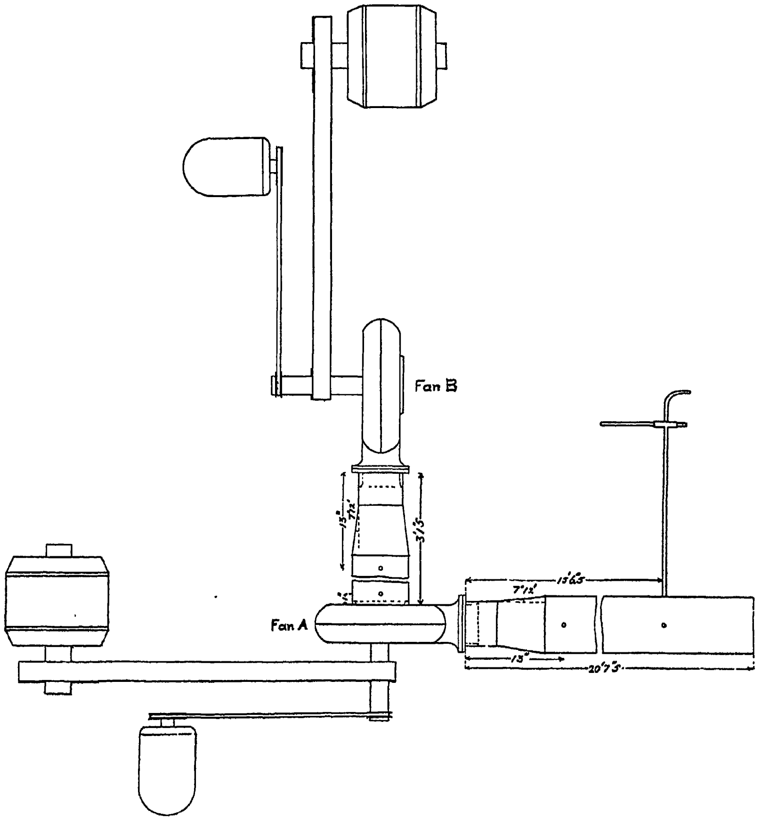


FIG. 3.—ARRANGEMENT OF APPARATUS FOR FANS IN SERIES.

CONDUCT OF THE TESTS

The fans were operated at the same speed in all tests, as shown by the electrical tachometers. The speed of both fans was 3500 r.p.m. Twelve points were taken in each traverse of the pipe for velocity. One man sat at the control table and held the fans at the desired speed while the other man read the manometer. The following tests were made—

1. Fan A alone with its expansion piece was placed on the duct and its static-pressure-volume characteristic was determined.

2. Fan B alone with its expansion piece was placed on the duct and its static-pressure-volume characteristic was determined.

Within the limits of precision of the instruments employed the characteristics of fans A and B were identical. Characteristics were determined only over the working range of the fans because of the chance of error in measuring lower velocities. In all characteristics the pressures are for a density of air of 0.075 lb. per cubic foot.

3. Fan B was set to blow directly into the inlet of fan A through a short piece of pipe as shown in Fig. 3.

The pressure produced by fan B was measured at its outlet at point shown. The pressure produced by fan A was measured between the inlet and outlet at points shown. The sum of the two fan pressures is equal to the drop in pressure in the outlet duct plus the drop in pressure in the connecting duct, in other words, is equal to the total duct resistance. A drop in pressure in the connecting duct was discernible only at the highest velocities.

Results

The characteristics of fan A or B operating alone are shown as the middle curve of Fig. 4, which shows the comparison of the individual fans when operating alone and in series.

The characteristic of fan A operating in series is higher than when the fan was operating alone, as would be expected, while the characteristic

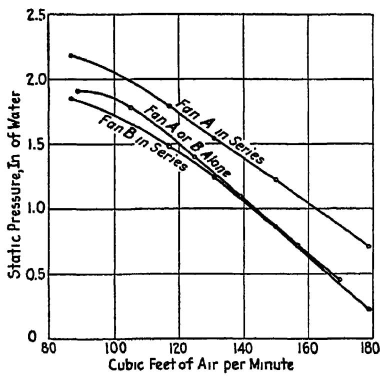


Fig. 4.—Fan characteristics.

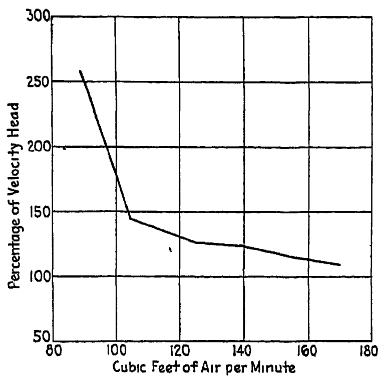


Fig. 5.—Gain in pressure in fan A in series.

of fan B is lower in series. The behavior of fan B in series indicates that some interference was caused by fan A, as possibly vibrations of the air which affected conversion in the expansion piece.

Fig. 5 shows the increase in static pressure of fan A in series plotted as a percentage of velocity-pressure at its inlet. The gain in pressure is in general more than the velocity-pressure supplied at inlet by fan B, and is greatest when the pressure at the inlet is highest. This indicates that filling the fan blades with air from another machine has a favorable effect on the operation of the fan.

The upper curve of Fig. 6 is the actual pressure-volume characteristic of the fans in series as determined by experiment. In all curves the points marked by a circle were determined by measurement both of pressure and volume.

The lower curve of Fig. 6 is a synthetic characteristic made by adding pressures at like volumes from the characteristics of fans A and B operated singly.

The excess of pressure of the actual characteristic over the synthetic characteristic amounts to a little more than the velocity pressure at the inlet of the forward fan over most of the range tested, so it would seem

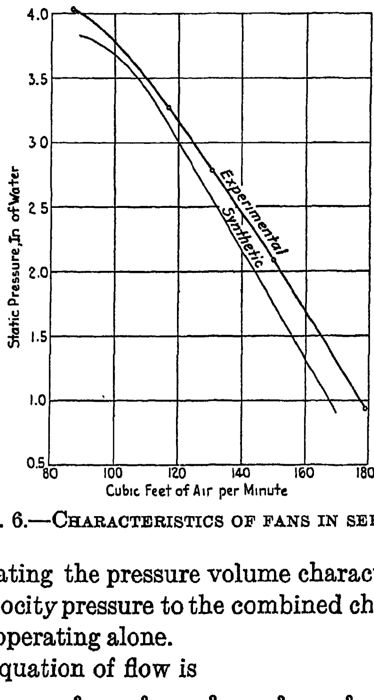


FIG. 6.—CHARACTERISTICS OF FANS IN SERIES.

reasonable in estimating the pressure volume characteristic of two fans in series to add this velocity pressure to the combined characteristic obtained from the fans when operating alone.

If this is done the equation of flow is

$$h_a + h_b + \frac{v_2^2}{2g} = \frac{v_4^2}{2g} - \frac{v_3^2}{2g} + \frac{v_2^2}{2g} - \frac{v_1^2}{2g} + f + s$$

Where h_a and h_b are the static heads of the fans when running alone, for the quantity involved, v_1 is the velocity at the outlet of the rear fan, v_2 is the velocity at the inlet of the front fan, v_3 is the velocity at the outlet of the front fan, v_4 is the velocity at the outlet of the forward duct, and s and f are the shock and friction losses in both ducts.

It is seen that $\frac{v_2^2}{2g}$ may be eliminated so the equation becomes

$$h_a + h_b = \frac{v_4^2}{2g} - \frac{v_3^2}{2g} - \frac{v_1^2}{2g} + f + s$$

A curve of the values of the left-hand side for different rates of flow is the synthetic static pressure characteristic of the fans, and a curve of the value of the right-hand side is the duct characteristic to use with it. The crossing of the curves indicates the rate of flow.

DISCUSSION

(*A. C. Callen presiding*)

G. E. McElroy, Reno, Nev. (written discussion).—The experiments reported in this paper practically duplicate, with smaller apparatus, those reported in U. S. Bureau of Mines *Technical Paper* 447 and confirm the results previously attained: (1) that the performance of a fan, on a total-pressure basis, is essentially the same regardless of whether it is used singly or in series; (2) that static pressure ratings, as given in fan manufacturer's catalogues, cannot be used to determine accurately the performance of fans in series. The writer prefers to consider fan performances and duct resistances on a total-pressure basis and thus avoid what are to him confusing conceptions. The authors of this paper, apparently, prefer to consider both on a static-pressure basis and give herewith the additional conception required for the particular case of fans in series.

Arsenic Elimination in the Reverberatory Refining of Native Copper*

By C. T. Eddy,† Houghton, Mich.

(New York Meeting, February, 1931)

The refining of native copper in the reverberatory furnace, as practiced in the Lake Superior district of Michigan, is very similar to the reverberatory melting and refining of cathodes, but the presence of arsenic in some of the copper concentrates renders the operation somewhat more difficult.

Though the copper occurs as the native metal, which is often exceptionally pure—purer, in fact, than after it has undergone refining—its intimate association with the gangue minerals necessitates the cumbersome reverberatory treatment. The gangue consists chiefly of complex silicates of calcium, magnesium, iron, aluminum and sodium, but in certain ore deposits other minerals are often found, among them the native copper arsenides—mohawkite, whitneyite, algodonite and the like. These latter minerals, when present, because of their relatively high specific gravity (7.5 to 8.5), find their way into the concentrates through the gravity concentration processes employed in the recovery of the copper. When such concentrates are melted, the silicates are removed as slag, but, because of the great affinity of molten copper for arsenic, the arsenides dissolve more or less completely in the copper bath. Because of the stability of the resulting melt, the arsenic is tenaciously held in solution.

The influence of arsenic on the properties of copper, especially on its electrical conductivity, is generally recognized. For certain uses it is beneficial, and in many instances its presence in limited amounts is not objectionable, but if the metal is to be used for electrical purposes, the almost complete removal of the arsenic is of prime importance.

Considerable research has been done on the subject of the influence of arsenic on the electrical conductivity of copper,[1] but the conclusions presented in the literature dealing with the problem show some disagreement. The methods now in use for the removal of arsenic have not

* The Alfred Noble Memorial Prize was awarded to the author for this paper.

† Assistant Professor of Metallurgy, Michigan College of Mining and Technology.

[1] D. Hanson and C. B. Maryat: Investigation of the Effects of Impurities on Copper. Part III. The Effect of Arsenic on Copper. Part IV. The Effect of Arsenic Plus Oxygen on Copper. *Jnl.* Inst. Met. (1927) **37**, 121–143.

L. Addicks: The Effect of Impurities on the Electrical Conductivity of Copper. *Trans.* A. I. M. E. (1906) **36**, 18.

F. L. Antisell: Relationship of Physical and Chemical Properties of Copper. *Trans.* A. I. M. E. (1920) **64**, 432.

been given the attention they deserve, nor have the reactions occurring in the process been the subject of careful study; therefore the purpose of the investigation set forth in this paper was to study these problems in detail.

Influence of Arsenic on Electrical Conductivity

The work on electrical conductivity was done with two purposes in view: first, to correlate the conductivity of arsenical copper in the cast condition with its conductivity in the hard-drawn and annealed states; and second, to aid in the study of the elimination of arsenic from the copper bath in the reverberatory refining of arsenical charges, for in the absence of other impurities the electrical conductivity furnishes the best and most rapid method for arriving at the arsenic content.

That the Lake Superior native copper was deposited in a high state of purity is evidenced by its unusually high electrical conductivity. Wires drawn from the original metal without melting have varied in conductivity from 101 to 103 per cent. when annealed, though the metal content (Cu + Ag) of such wires may range from 99.60 to 99.99 per cent., depending on the amount of mechanically held gangue particles. In this connection the following comparisons may be interesting. Commercial copper wire bars, with a purity of from 99.94 to 99.96 per cent. (Cu + Ag), average from 100 to 101 per cent. conductivity. Fire-refined lake copper drillings, freed from sulfur and oxygen by ignition in hydrogen, and melted in a vacuum in an induction furnace, had a purity of 99.98 per cent. and a conductivity of 101.5. Heuer[2] reports that copper of the highest purity melted in the most careful manner shows a conductivity of over 102 per cent. His analyses indicate a purity closely approaching 100 per cent.

The copper produced by melting Lake Superior native copper concentrates usually has a lower conductivity than the pure metal, because of the arsenides in the gangue, as previously explained. The amount of arsenic thus introduced depends on the origin of the material, but it is generally recognized that, except for a slight lowering of the conductivity due to silver, arsenic is the only impurity affecting the electrical properties to any great extent.

To determine accurately the effect of arsenic on the conductivity of lake copper, a series of melts was made in the following manner. Refined lake copper[3] was melted in an induction furnace in a clay-graphite crucible under a cover of borax. High-arsenic copper (2.0 to 2.5 per cent. As) was then added in the proper proportion and the melt, after

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[2] R. P. Heuer: The Effect of Iron and Oxygen on the Electrical Conductivity of Copper. *Jnl.* Amer. Chem. Soc. (1927) 49, 2711–20.

[3] Analysis: Cu + Ag- 99.95 per cent.; oxygen, 0.043; silver, 0.016: arsenic, 0.0030 per cent. Electrical conductivity, 100.3 per cent.

being stirred, was cast in "spike bar" molds. Four spikes, approximately ½ × ½ × 8 in., were made for each melt. The majority of the charges were of approximately 1000 grams, though for each variation in arsenic of 0.25 per cent. an additional melt of 2000 g. was made. The larger melts were cast in a vertical billet mold and were used for microscopic examination and physical testing. In general, an attempt was made to produce test specimens for every 0.1 per cent. variation in arsenic from 0.1 to 1.0 per cent., and at much more frequent intervals in the range below 0.1 per cent.

For the conductivity determinations, spikes were heated to 800° C. and hot-rolled to 5⁄16-in. rods; the rods were annealed at 800° C. and drawn in seven passes to 0.102-in. dia. wire (No. 10 B. & S. gage). The resulting wires with arsenic contents below 0.4 per cent. were tested in a

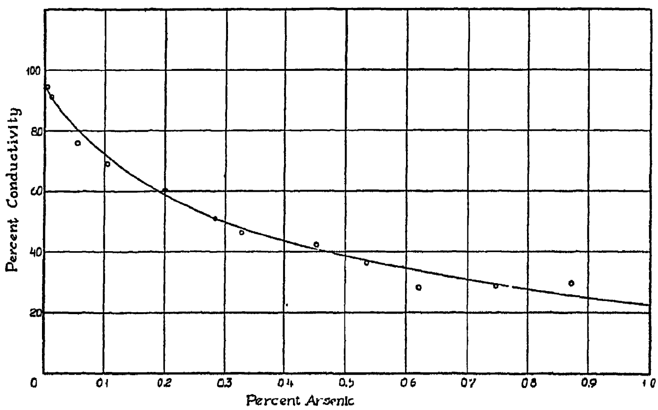

Fig. 1.—Arsenic-conductivity relation for cast copper.

Hoopes conductivity bridge before and after annealing at 800° C. A Leeds & Northrup Kelvin bridge was used for the higher arsenic wires which were beyond the range of the Hoopes bridge. One spike from each lot was machined to a 3⁄16-in. square section and its conductivity was determined with a Leeds & Northrup type K potentiometer by the drop-in-potential method.

The densities of the cast metal were determined by weighing a machined section of one of the spikes in air and water. Temperature corrections necessary for arriving at the true volume of water displaced were applied.

Arsenic percentages were found by a modified Gutzeit method, developed at the laboratory, which was both rapid and accurate. Frequent check analyses were made by the standard distillation method.

In Table 1 are listed the conductivities of arsenical copper varying from 0.0025 to 1.0 per cent. arsenic, in the cast, hard-drawn and annealed states.

Fig. 1 shows the conductivity of the cast metal plotted against percentage of arsenic. The conductivity of the cast bar is somewhat lower than that of the hard-drawn and annealed wires of the same analysis. This difference appears to be due to the density differences, and is relatively consistent with changes in the arsenic content. The difference between the conductivity of the annealed wire and of the cast bar at 0.0025 per cent. arsenic is approximately 5.0 per cent., and seems to decrease slightly with increasing arsenic. At 1.0 per cent. arsenic, the difference was reduced to 2.5 per cent. Annealing the cast bar decreased its conductivity somewhat, especially in the higher arsenic bars.

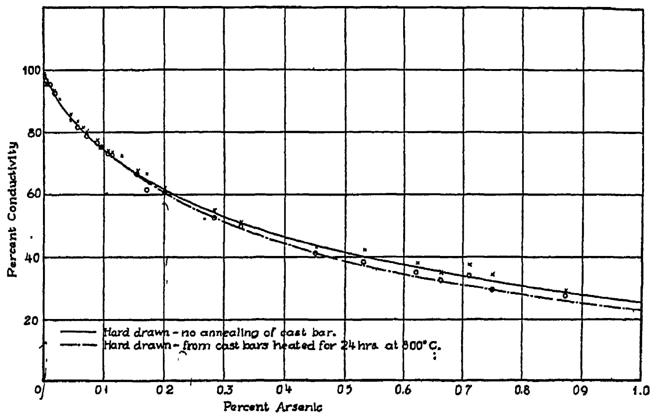

FIG. 2.—ARSENIC-CONDUCTIVITY RELATION FOR HARD-DRAWN WIRES.

In Fig. 2 the results for the hard-drawn wires are given, plotted against arsenic contents as before. The conductivity of the hard wire, as is shown by the full-line curve, was higher than that of the annealed wire for arsenic contents above 0.11 per cent., the difference becoming greater with increasing arsenic. The tendency toward higher conductivity in the hard-drawn wires decreased when the bars were heated for long periods before rolling and drawing. The curve representing the conductivities of hard wires, secured from a set of determinations run on bars which had been heated for 24 hr. at 800° C., is practically coincidental with the curve representing the annealed wires. The former curve is shown as a broken line on Fig. 2 and again in Fig. 4.

The curve for annealed wires is shown in Fig. 3. The results plotted are for the unheated spikes. The wires from the heated spikes gave conductivities closely approaching the others, though with increasing arsenic there was a slight decrease from the values shown in the curve.

TABLE 1.—*Relation Between Electrical Conductivity and Arsenic Content*

As, Per Cent.	Electrical Conductivity, Per Cent.				
	As Cast	Cast Bar Heated to Rolling Temp. and Rolled Immediately 800° C.		Cast Bar Heated to 800° C. and Annealed at This Temp. for 24 Hr.	
		Hard	Annealed	Hard	Annealed
0.0025	95.2	98.4	100.5	98.4	100.5
	94.4	97.45	99.85		
		98.70	100.65		
		97.55	99.9	97.4	100.0
0.0050		96.5	98.6		
		97.1	99.25	96.6	99.1
0.0075		95.75	97.9	95.1	97.3
		95.6	97.5		
0.0080		95.0	97.1		
		95.6	97.3		
0.009	91.3	94.1	96.2	94.2	96.5
		94.8	96.7		
0.010		94.7	96.5	94.3	96.2
0.016		93.4	95.6	92.2	94.1
0.025		90.3	91.2	90.2	91.0
		90.85			
0.044		85.75	87.2	85.4	87.3
		83.9	86.1		
		85.0	86.9		
0.058	76.1	82.8	83.75	81.9	83.5
0.065		81.5	82.2		
0.073		80.0	80.5	78.5	79.25
0.090		77.5	77.9	76.1	78.0
0.098		75.5	75.7	75.6	75.9
0.106	69.75	74.0	74.0	73.7	74.1
0.112		73.5	73.5	73.0	73.3
0.125		72.1	71.8		
0.131		72.1	70.7		
0.155		67.7	67.5	66.6	66.8
0.172		66.4	65.3	61.9	62.2
0.200	60.2	61.5	60.6	60.5	60.3
0.285	51.1	54.0	52.3	52.5	52.1
0.330	46.2	51.3	49.4	50.0	49.3
0.455	42.1	42.4	41.5	41.0	41.0
0.534		40.0	37.9	38.2	37.9
0.623	28.7				
		38.1	35.2	34.7	34.3
0.665		35.0	32.6	32.5	32.1
0.712		33.6	31.3	34.0	33.3
0.750	29.2	34.2 1-4" Rod	30.4	29.5	29.5
0.875	29.4	28.8 1-4" Rod	26.7	27.5 1-4" Rod	26.9 1-4" Rod

Fig. 4 shows the relationship of the various curves to each other; the individual points representing the data are omitted. The curves $A'B'$ and AB' for both the hard and the soft wires drawn from bars

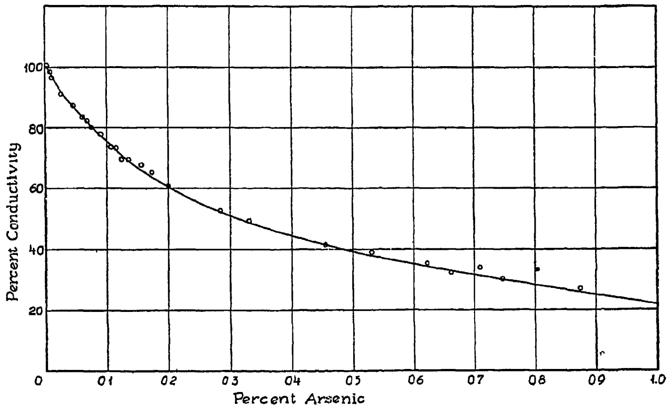

FIG. 3.—ARSENIC-CONDUCTIVITY RELATION FOR ANNEALED WIRES.

heated for 24 hr. indicate that with increasing arsenic the conductivities become practically identical.

The cast arsenical copper showed a tendency toward cored crystals and segregation, a condition which was removed by the long heating

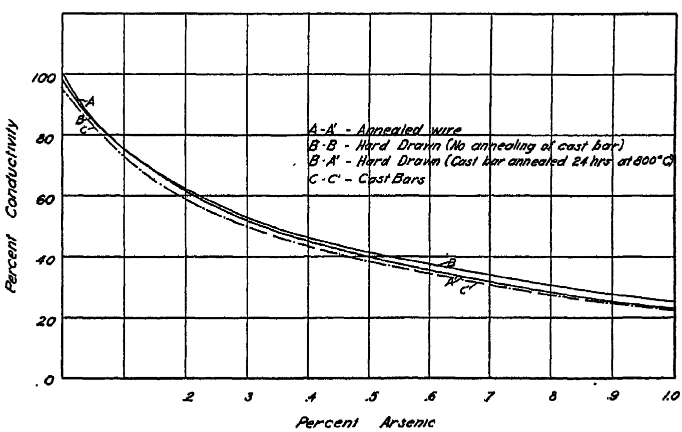

FIG. 4.—RELATIONSHIP OF ARSENIC-CONDUCTIVITY CURVES FOR CAST COPPER, HARD-DRAWN AND ANNEALED WIRES.

of the cast bar. This removal as a result of heating accounts for the drop in the conductivity of both the cast bar and the hard wire.

With the exception of the range from apparently pure copper to 0.1 per cent. arsenic, the electrical conductivity of the drawn and the

annealed metal varies approximately as the logarithm of the arsenic content; *viz.*,

$$\text{Per cent. conductivity} = 240.4 - 54.5 \log. \text{As},$$

the arsenic being expressed in parts per million. This relation is obviously untenable in the range from arsenic-free to 0.10 per cent. arsenic, where other influences are exerted which probably do not affect the higher arsenic range to any appreciable extent.[4]

Arsenic exists (in copper) as a solid solution with the copper in concentrations as great as 7 per cent.,[5] though a part is thought to be associated with the cuprous oxide[6] which is always present in tough-pitch copper, particularly when the arsenic is present in amounts over 0.3 per cent. Variation in the oxygen content up to 0.1 per cent. did not exert much influence on the conductivity of the arsenical wires. The conductivity of the material with low arsenic content was improved slightly by an increase in the oxygen to 0.05 per cent. and then was lowered again as the oxygen increased beyond this value. With oxygen contents over 0.05 per cent., the conductivity varies as a linear function of the oxygen for any one arsenic content between 0.01 and 0.88 per cent.

In refining the samples, no attempt was made to reduce the oxygen content below 0.05 per cent., because, especially with the higher arsenic melts, it was possible to secure castings that were much more nearly sound and of higher density than those that would result if the metal were deoxidized to the fullest extent. Furthermore, the purpose being to study arsenical copper as made in the reverberatory furnace, the usual

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[4] The research department of the Calumet and Hecla Consolidated Copper Co. has derived equations connecting the electrical conductivity and the arsenic content by means of the theory of series circuits, as follows:

For lake copper containing no oxygen,

$$\text{Per cent. conductivity} = \frac{1}{0.009878 + 0.02729\ \text{As}\ (\%)}$$

For tough-pitch lake copper, for the range from 0.17 to 0.60 arsenic,

$$\text{Per cent. conductivity} = \frac{1}{0.010295 + 0.03186\ \text{As}}$$

and for the range from zero to 0.17 arsenic,

$$\text{Per cent. conductivity} = \frac{1}{0.010295 + 0.03186\ \text{As}} + 3.365 - 19.45\ \text{As}.$$

It developed from these equations that the abnormal lowering of the conductivity in the lower ranges was due to varying amounts of either As_2O_3 or else As_2O_5 in solid solution. Beyond 0.17 per cent., the amount of the oxide in solid solution became constant.

[5] D. Hanson and C. B. Maryat: *Op. cit.*

[6] J. Rhurmann: Über Arsen und Nickel sowie deren Sauerstoffverbindungen im Kupfer und ihren Einfluss in geringen Mengen auf seine mechanischen Eigenschaften. *Metall u. Erz* (1925) 22, 339–348.

oxygen content of tough-pitch copper was maintained as closely as possible. The oxygen percentage did not run below 0.04, and only in the last few high-arsenic samples was the oxygen present in amounts higher than 0.06, and even then never above 0.085 per cent.

Arsenic Elimination

The elimination of arsenic from the copper bath in the reverberatory furnace normally takes place at the end of the oxidizing period after the charge has been freed from sulfur[7] and the oxygen content is approximately 1.0 per cent. At this time soda ash (commercial sodium carbonate) is injected into the bath with air, forming a fluid slag, and the arsenic enters the slag. After a sufficient quantity of soda has been introduced, and the soda slag containing the arsenic has been removed by skimming, the charge is poled to pitch in the usual manner. Fig. 5 shows graphically the extent of the arsenic elimination during the refining of two typical furnace charges of arsenical copper. The abrupt drop in the arsenic analysis may be noted as starting at the point where the blowing of the soda began. In the charge represented by the full lines, the soda was introduced when the oxygen content of the bath was 1.05 per cent., and in the other the oxygen had been lowered by poling before the introduction of the soda.

Arsenic and copper are miscible in all proportions in the liquid state; however, it has been demonstrated[8] that in the presence of oxygen the arsenic forms a copper arsenate according to the following reactions:

$$5Cu_2O + 2As \rightleftarrows As_2O_5 + 10Cu \qquad [1]$$

$$XAs_2O_5 + Cu_2O \rightleftarrows (Cu_2O)X(As_2O_5) \qquad [2]$$

The copper arsenate which is formed is insoluble in the liquid melt. These results, as far as they have been checked in our laboratories, have been confirmed; however, our tests show that unless arsenic is present in relatively large amounts (0.8 per cent. or over), the formation of the arsenate exerts little influence on the elimination of the arsenic by soda ash.

The reaction by which arsenic elimination takes place is probably represented by the following expression:

$$4As + 5O_2 + 6Na_2CO_3 = 4Na_3AsO_4 + 6CO_2 \qquad [3]$$

or perhaps more accurately by the following reactions:

$$2As + 5Cu_2O = 10Cu + As_2O_5 \qquad [4]$$

$$As_2O_5 + 3Na_2CO_3 = 2Na_3AsO_4 + 3CO_2 \qquad [5]$$

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[7] H. O. Hofman: Metallurgy of Copper, Ed. 2. New York, 1924. McGraw-Hill Book Co.

[8] J. Rhurmann: *Op. cit.*

The evidence is very strong that the arsenic must exist as the higher oxide in order that its elimination may proceed. It is always found in the pentoxide condition in the soda slag.

It might be expected that the formation of copper arsenate according to equations 1 and 2 would aid elimination and possibly decrease the

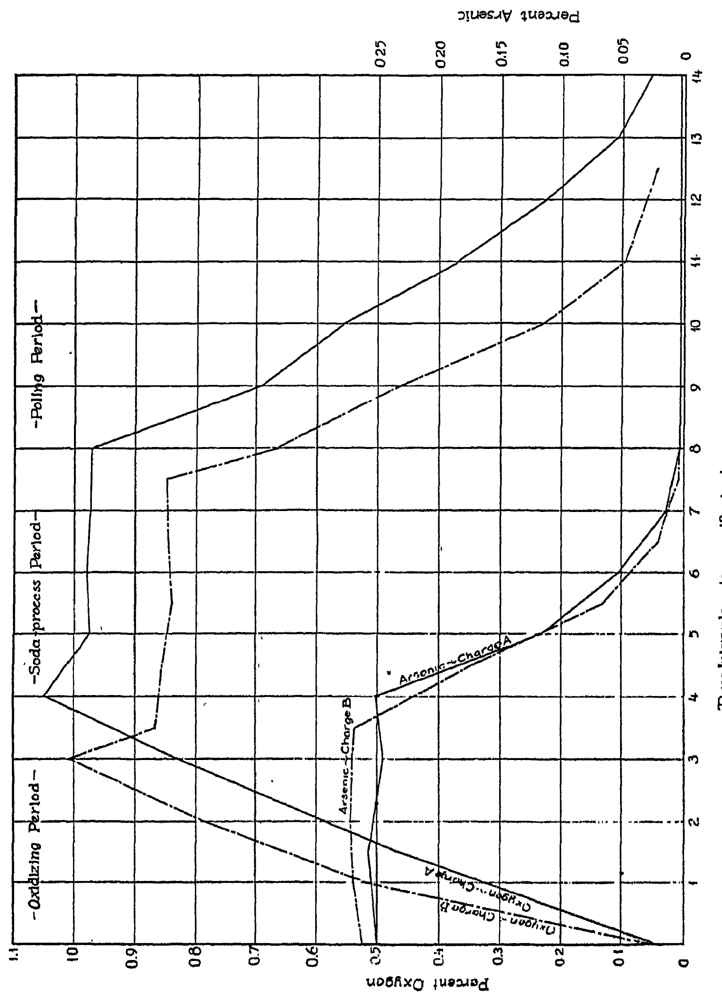

FIG. 5.—GRAPHIC REPRESENTATION OF TWO TYPICAL FURNACE CHARGES.

soda consumption, but it will be shown that the elimination takes place more efficiently with moderate concentrations of oxygen than with the high concentrations necessary for arsenate formation. For example, in charge *B* in Fig. 5, the soda consumption was lower than in charge *A*, although there was less oxygen present in the former case.

It may be observed in Fig. 5 that there is a tendency for the oxygen content of the bath to decrease as the arsenic is eliminated. This decrease is to be expected from the reactions, not only because the elimination of the arsenic presupposes the elimination of oxygen as arsenic pentoxide, but also because of the solubility of cuprous oxide in molten sodium carbonate. Notwithstanding the slow rate at which molten copper containing over 1 per cent. oxygen will absorb more, the drop in oxygen content with the introduction of soda was often difficult to find, much more to determine.

Considerable experimentation was done in a 25-lb. crucible in an oil fired furnace. In these experiments, the drop in the oxygen content incident to the addition of the soda could not be observed unless the soda was thrown on top of the melt all at one time; never if the soda was injected with air as in the usual procedure. Naturally, if the soda is not injected, the arsenic elimination will be much slower and the slag will be richer in cuprous oxide.

Most of the experimental work herein reported was carried out in a 2-ton reverberatory furnace in our laboratory.[9] In this work, the drop in the oxygen content with the introduction of soda was almost as difficult to trace as in the tests involving the use of the crucible, unless precautions were taken to introduce as little air as possible with the soda and to keep the furnace openings as small as possible. In some cases where these precautions were not taken, the oxygen content actually increased.

The constitution of the soda slag is a problem in itself. The equilibrium relations involved between the concentrations of the various phases both in the metal bath and in the slag and the influence of the temperature on the equilibrium are now being studied and will be discussed in a later paper.[10] Since there is no known refractory that will withstand the corrosive action of molten sodium carbonate at the temperature of molten copper for even a moderate length of time, the difficulties involved can be easily understood. The chief drawbacks to the use of soda in fire-refining are, of course, the high cost of the necessary refractories, the frequency with which they must be renewed, and the high furnace repair costs involved.

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[9] For a description of this furnace, see C. T. Eddy: A Two-ton Experimental Reverberatory Furnace at the Michigan College of Mining and Technology. *Proc.* Lake Superior Min. Inst. (1929) **27**, 67–75.

[10] The Calumet and Hecla research department has found that the system molten copper-soda slag behaves as a liquid-liquid system, and the distribution ratio, arsenic in the slag to arsenic in the metal, is something over 500 to 1, provided that the slag layer is not saturated. If this is true the As_2O_5 should be soluble in the metal bath up to the usual concentrations. The research department states that the theoretical ratio is not reached in practical operation but that 70 per cent. of the theoretical figure is good practice.

Arsenic is seldom present in lake copper in amounts greater than 0.3 per cent. In this proportion it appears to be readily soluble in the metal in the liquid state even when the oxygen content is as high as 1 per cent. In the solid condition, arsenic in amounts less than 0.3 per cent. does not appear to affect the microstructure. In samples with low oxygen content no difference was observed. With increasing oxygen a tendency to columnar structure and smaller grain size was noticed, though the appearance of the Cu_2O dendrites and the Cu_2O in the eutectic was practically identical in samples of 0.0025 and 0.28 per cent. arsenic.

Soda Consumption

In addition to the arsenic content and the temperature of the bath, the chief factors that influence the soda consumption are the Cu_2O content and the mode of introduction of the soda.

In Fig. 5 two typical furnace charges are followed through graphically and the Cu_2O and arsenic concentrations are shown during the various stages of the refining process. In Fig. 6 the soda consumption for these two charges is given, also graphically. In order to reduce the figures to a definite standard, the soda consumption for 100 lb. of copper in the bath is used as a basis of comparison. These results are set forth in detail in Table 2.

Table 2.—*Soda Consumption for Charges A and B*

Charge A (3750 lb.)		Charge B (1) (3810 lb.)	
As, Per Cent.	Soda[a]	As, Per Cent.	Soda[a]
0.250	none	0.261	none
0.1620	0.58	0.172	0.52
0.1175	1.17	0.110	1.05
0.0833	1.76	0.078	1.57
0.0510	2.35	0.0485	2.10
0.0375	2.94	0.0255	2.62
0.0210	3.52	0.0173	3.15
0.0105	4.11	0.0080	3.67
0.0031	4.70	0.0025	4.20
176 lb. soda used		160 lb. soda used	

[a] Soda = pounds per 100 lb. of bath.

From reaction 3 it may be calculated that, to form Na_3AsO_4, 2.2 lb. of sodium carbonate is required for each pound of arsenic in the bath. The curves show that the amount actually used was much greater than this quantity, and also that the consumption was greater in charge *A* (Figs. 5 and 6), where the soda was blown into a bath of 1.0 per cent. oxygen, than in charge *B*, in which the soda was introduced with the

oxygen at 0.86 per cent. Other influences, such as the time factor, may have accounted for a part of the increase, but it has been found that if the best efficiency is to be obtained there is a rather sharp limit to which the

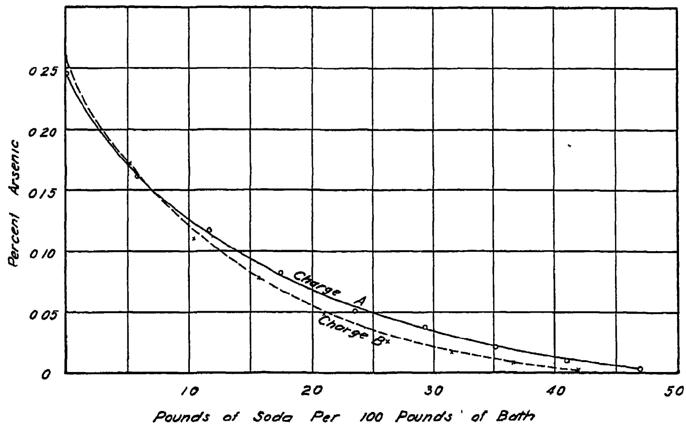

FIG. 6.—SODA CONSUMPTION FOR CHARGES REPRESENTED IN FIG. 5.

bath may be deoxidized before the addition of the soda. For the purpose of determining the optimum conditions, deoxidation by poling was carried out in varying degrees before soda was introduced. From the data

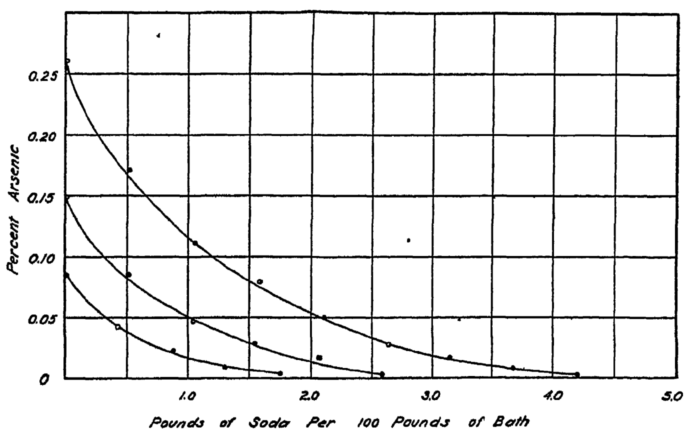

FIG. 7.—SODA CONSUMPTION FOR CHARGES OF 0.085, 0.15 AND 0.26 INITIAL PERCENTAGE ARSENIC, RESPECTIVELY.

obtained, it was determined that the oxygen content resulting in the lowest soda consumption in the refining of a bath containing 0.3 per cent. arsenic was approximately 0.85 per cent.; and that this value did not change materially as the arsenic content of the bath decreased. The

effect on the soda losses of too little Cu_2O in the copper is more pronounced than that brought about by a high Cu_2O content up to 1.1 per cent. If the amount of Cu_2O is too low, the oxygen content works up gradually during the blowing of the soda; but a longer time is required to eliminate the arsenic, the copper losses in the slag increase, and the soda consumption is greater.

In Table 3 are listed the results obtained from the sampling of three charges, of 0.085, 0.15 and 0.26 per cent. arsenic respectively. The results are presented graphically in Fig. 7, the 100-lb. basis being used, as before. The oxygen contents of these charges varied between 0.85 and 0.91 per cent.; the electrical conductivity of the finished product was 99.5 to 100.0 per cent.

TABLE 3.—*Soda Consumption for Three Charges of Different Arsenic Contents*

Charge 1 (B) (3810 lb.)		Charge 2 (3880 lb.)		Charge 3 (3680 lb.)		Remarks
As, Per Cent.	Soda[a]	As, Per Cent.	Soda[a]	As, Per Cent.	Soda[a]	
0.261	none	0.149	none	0.085	none	Initial arsenic
0.172	0.525	0.0850	0.515	0.044	0.43	
0.110	1.050	0.0475	1.030	0.023	0.87	
0.078	1.575	0.0280	1.545	0.009	1.30	
0.0486	2.100	0.0165	2.060	0.003	1.74	
0.0255	2.625	0.0022	2.575			
0.0173	3.150					
0.0080	3.675					
0.0025	4.200					
160		100		64		Pounds of soda
16.10		17.25		20.47		Pounds of soda per pound of arsenic

[a] Pounds of soda = pounds per 100 lb. for bath.

For the sampling of these charges, the addition of the soda was stopped temporarily, the bath was stirred by hand for a moment, and a 25-lb. ladle of metal was dipped from the furnace. From this ladle the test bars were cast; the remaining copper in the ladle was run back into the bath.

By plotting the values for the number of pounds of soda per pound of arsenic, given at the bottom of Table 3, and extrapolating the curve, it becomes possible to estimate the probable soda consumption per pound of arsenic for any initial arsenic content from 0.4 per cent. to zero..

Summary

The origin of the arsenic in Lake Superior copper is traced and its effect on the properties of the metal is described.

A comparative study is given of the effect of arsenic on the electrical conductivity of lake copper in the cast, hard-drawn and annealed states. Because of segregation in the cast bar, the test wires containing more than 0.10 per cent. arsenic give higher conductivities when hard drawn than after annealing. Prolonged heating of the cast bar overcomes this tendency, so that the curves representing the hard-drawn and annealed states become practically contiguous.

The electrical conductivity is a straight-line function of the logarithm of the arsenic content in the range 0.10 to 1.0 per cent. arsenic, but, because of the complexity of the relationship, no equation tenable for the whole range from 0.0 to 1.0 per cent. arsenic is attempted.

Data relevant to arsenic elimination by the soda-ash method are given, and the soda consumption is computed. It is shown that important influences are exerted, not only by the temperature of the bath, but also by the concentration of the oxygen and by the mode of introduction of the soda. The effect of varying concentrations of oxygen is illustrated, and the optimum oxygen concentration deduced. An oxygen content of 0.85 per cent. results in the least soda consumption. The formation of copper arsenate is shown to have little effect on the arsenic elimination, but all of the arsenic must be present in the pentavalent condition.

Data are given to illustrate the relationship between the initial arsenic concentration in the bath and the amount of soda required for its removal; from these data, the probable soda consumption for any furnace charge containing any amount of arsenic from a very little to 0.4 per cent. may be estimated.

Acknowledgments

The writer wishes to express appreciation for the cooperation received from the research department of the Calumet and Hecla Consolidated Copper Co. Acknowledgment is also made to Mr. S. Skowronski of the Raritan Copper Works, Perth Amboy, N. J., for his suggestions in the preparation of the paper, to the Michigan Smelting Co. and to the Quincy Smelting Co. for their cooperation in obtaining and checking some of the data.

DISCUSSION

(Carle R. Hayward presiding)

G. P. Schubert, Houghton, Mich. (written discussion).—In the practical application of soda treatment for arsenic elimination with furnace charges of 250,000 to 450,000 lb. of refined copper, it has been noted that theory and practice agree closely

in a chemical and metallurgical way. The efficiency of the soda treatment of high-arsenic charges (As from 0.2 to 0.4 per cent) is high at the beginning of the reduction, but decreases quickly as the higher conductivities (98 to 100 per cent) are attained.

The phase of treatment which presents the greatest difficulty in its practical application is the fact that while arsenic can be actually and readily removed to a low percentage in the bath (0.005 per cent or less), yet when the level of the bath is lowered during the casting period certain contamination takes place, the speed and intensity of this contamination depending upon the porosity of the side walls, welts and bottom of the reverberatory refining furnace, as well as on the drips of arsenical slag and copper which may have accumulated during the rabbling, soda blowing with compressed air, and poling, all of which create violent agitation and splashing. Contamination from fugitive arsenic in bottom and welts could be minimized were it possible to build nonporous and stable side welts and bottoms of relatively inexpensive basic materials.

In practice, where the products of arsenical and soda-refined copper are produced in alternate charges, in the same furnace, the soda consumption per ton of refined copper is apt to be as high as 70 lb. or more (3.5 lb. per 100 lb. copper). The introduction of so much basic material under the necessary heat conditions is most damaging to any of the known and practical furnace linings.

The manual manipulation of soda-blowing pipes, rabble pipes, rabbles, quick and complete removal of resulting soda slag, appears to be most important, so that the success of the treatment depends greatly upon the skill and efficiency of the furnace workers.

High-silica Retorts at the Rose Lake Smelter

By G. L. Spencer, Jr.,* East St. Louis, Ill.

(New York Meeting, February, 1931)

There is no question as to the importance of the part played by the retort in modern zinc smelting. A satisfactory retort should have properties that will result in resistance to slagging action and fire cracks, and have no tendency, or at least only a small tendency, to soften and bend under furnace heat. The life and cost of a retort also are extremely important factors. The trend in zinc smelting during the past few years has been toward longer retorts and higher charge density. This is caused by the fact that the diameter is limited to 9 or 10 in. because of heat penetration difficulties in larger sizes. Consequently, increasing the length of the retort and density of charge offers the only practical means of increasing the amount of zinc produced per retort. The stiffness of a retort perhaps is the most important factor, as slag accumulates in the bottom of a retort which has sagged and very quickly cuts through, often ruining the retorts below. The development of the high-silica retort is a result of the demands made by the trend toward heavy charges.

Materials Used for High-silica Retorts

The materials used in the manufacture of high-silica retorts at East St. Louis are fireclay, grog and silica flour. Analyses are given in Table 1. The fireclay comes from the St. Louis district. The grog consists of clean firebrick culls, and the silica flour, a very high-grade material, is purchased in the form of a flour that has been ground through 140 mesh.

Table 1.—*Analysis of Materials Used in High-silica Retorts*

Material	Ignition Loss, Per Cent.	Silica, Per Cent.	Alumina, Per Cent.	Ferric Oxide, Per Cent.	Lime, Per Cent.	Magnesia, Per Cent.
Fireclay	10.23	60.62	23.73	3.63	0.77	0.66
Grog	0.00	57.25	38.00	3.50	0.77	0.63
Silica flour	0.11	99.62		$Fe_2O_3 + CaO$ 0.26		

The clay is allowed to weather in open bins, then is broken up on a 14 by 18-in. crank and toggle type jaw crusher and elevated to a trommel

* Assistant Chief Metallurgist, American Zinc, Lead & Smelting Co.

screen which has 5-mesh wire cloth, No. 14 wire. The undersize is sent to the working storage bins while the oversize is delivered to a pair of rolls, each 14-in. face and 24-in. dia., which are in a closed circuit with the trommel screen. A screen analysis of this clay is given in Table 2. The grog is prepared by crushing the cull brick with the same equipment as is used for the clay. A screen test on this material is also given in Table 2.

TABLE 2.—*Screen Analyses of Clay and Grog*

Screen Size,[a] Mesh	Fireclay		Grog	
	Total, Per Cent.	Cumulative, Per Cent.	Total, Per Cent.	Cumulative, Per Cent.
On 3	0.00	0.00	0.00	0.00
6	0.03	0.03	0.29	0.29
10	33.43	33.46	42.99	43.28
20	28.86	62.32	27.60	70.88
35	16.48	78.80	15.21	86.09
65	9.64	88.44	7.13	93.22
100	5.23	93.67	2.60	95.82
150	2.29	95.96	1.07	96.89
200	1.58	97.54	0.81	97.70
Through 200	2.45	100.00	2.30	100.00

[a]All screen tests based on Tyler standard sieve scale.

MANUFACTURE OF RETORTS

The materials, now ready for the first mixing, are fed by hand into a No. 30 low-frame combination pug mill and fill machine with a No. 75 pug pan, 23 in. by 8 ft., made by the American Clay Machinery Co. Since the quantities of materials used are based on volume percentages, buckets of definite sizes are used to measure the amount of each material used. The three materials are mixed in the following proportions by volume: clay 50 per cent., silica flour 25 per cent., grog 25 per cent. In addition, water equivalent to give a moisture content of 10 to 12 per cent. is added in this mill. The product is an endless 8 by 8-in. ballot, which is cut into 2-ft. lengths by hand.

These ballots are stacked on the floor and after a number have been made are repugged through the same machine, in order to get as uniform a mix as possible. After the second pugging the material is covered with moist burlap and stored in a tempering room for approximately 10 days. During this time the mix becomes more plastic, owing to the dissemination of moisture and the action of the colloidal materials in the clay. An actual analysis of the retort mix is: ignition loss, 5.59 per cent.; silica, 66.06; alumina, 24.22; ferric oxide, 2.96; lime, 0.56; magnesia, 0.52.

After tempering, the ballots are put through an International Clay Machinery C. W. R. 6–0 brick machine which is double geared, with a 5¾-in. dia. auger shaft. This operation forms the mud into a cylindrical ballot of 14-in. dia., which is cut off at intervals, each cutting furnishing just enough mud for one retort. These ballots are then pressed into shape in a Wettengel hydraulic press.[1]

Five men are required in the process of forming the retorts, one press man, two ballot-mill men and two retort truckers. After the retort is pushed up and out of the press, it is cut off to the desired length and the mouth is smoothed with a hardwood block. It is then trucked to the drying rooms. Retorts are dried at room temperature for 10 days, then the temperature is raised to 128° or 130° F. and the retorts dried for 60 to 90 days. Losses in the drying process amount to 5 per cent.

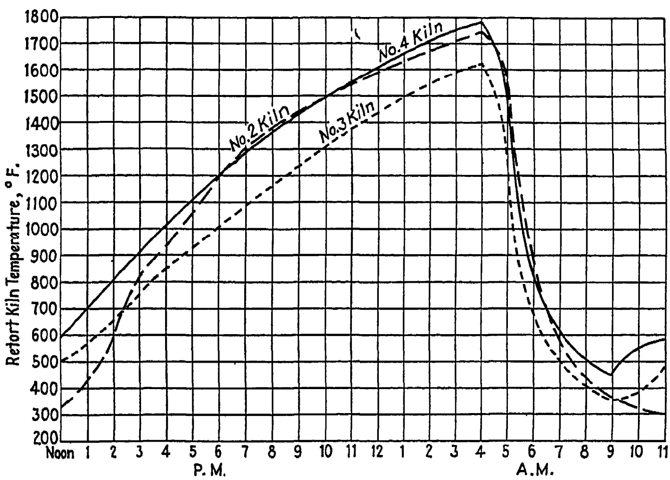

FIG. 1.—FIRING CURVE FOR HIGH-SILICA RETORTS. AVERAGE, JUNE 1–10, 1930. NATURAL-GAS FIRING.

The dried retorts are taken as needed from the drying room and transported by truck to the kilns on the furnace floor, which are fired by natural gas. The kilns are loaded and sealed; the firing is started at about 11 o'clock in the morning and lasts until 4 o'clock the next morning. The firing operation is extremely important. Fig. 1 is a reproduction of an actual firing curve.

If the temperature is brought up too rapidly or allowed to fall at any time during the operation prior to the end, strains are set up in the retort which cause early failure in the furnace. It is also very important that the retort be taken from the kiln and placed in the furnace as quickly as

[1] For a complete description of this press see H. O. Hofman: Metallurgy of Zinc and Cadmium, 145–148. New York, 1922. McGraw-Hill Book Co.

possible, as a material drop in temperature at this point is also detrimental to the life of the retort. Losses in the preheating kilns amount to 3 per cent.

Service Given by Retorts

With the high-silica mix, it is possible to use longer retorts than with the ordinary clay mix, because of the superior strength of the former. Failures due to slag troubles are materially reduced also, since there is no pocket formed, due to sagging, in which slag can accumulate. This makes for easier cleaning, as most of the slag accumulates at the retort mouth. Table 3 shows the average life of high-silica retorts by rows for one year.

Table 3.—*Average Life of High-silica Retorts*

1929 Month	Row 1	Avg. Life	Row 2	Avg. Life	Row 3	Avg. Life	Row 4	Avg. Life	Row 5	Avg. Life	Row 6	Avg. Life	Total	Avg. Life
Jan.	1,037	19.9	753	26.7	571	36.7	496	40.0	488	39.9	117	49.8	3,462	30.9
Feb.	845	20.1	628	28.0	451	37.6	391	41.3	415	40.3	97	48.3	2,827	31.5
March	899	22.7	640	29.8	519	40.9	409	48.0	466	43.7	111	43.9	3,044	34.0
April	816	23.1	584	32.3	397	42.6	463	51.4	469	43.8	111	42.9	2,840	36.3
May	773	24.5	587	33.9	457	43.4	423	43.9	482	40.0	112	44.2	2,834	35.3
June	684	27.9	580	34.6	479	45.9	390	48.1	458	41.8	103	45.6	2,694	37.
July	781	26.8	593	33.4	432	41.5	349	51.2	391	46.8	109	44.0	2,656	36.8
August	716	26.2	634	32.2	491	42.2	361	54.0	381	47.9	111	45.5	2,694	37.4
Sept.	541	28.5	495	32.5	401	41.7	315	53.6	339	49.8	109	48.6	2,200	39.2
October	586	27.3	485	29.9	429	36.3	307	40.1	340	43.0	100	40.2	2,253	35.9
November	838	25.2	639	31.0	545	36.0	437	44.0	353	45.5	105	45.7	2,917	34.1
December	747	25.1	625	30.7	549	35.0	440	45.5	428	53.8	78	48.9	2,867	35.4
Total and average	9,263	24.8	7,243	31.3	5,721	39.9	4,781	46.8	5,010	44.7	1,269	45.6	33,288	35.2

No definite conclusions have been reached as to the reason why the addition of silica flour to the retort mix results in a superior product. It may be due to the colloidal nature of the silica flour; that is, the gel formed during the tempering envelopes all particles of clay and grog and when burned gives rise to a monolithic structure of extreme stiffness.

The service obtained from high-silica retorts varies with the type of furnace in which they are used, and with the method of firing. At the Rose Lake smelter Siemens Neureuther furnaces are used entirely and are fired to give maximum production of metal per retort.

DISCUSSION

(*Frank G. Breyer presiding*)

E. M. Johnson, Henryetta, Okla. (written discussion).—We make our retorts in about the manner described by Mr. Spencer, but we do not let the pugged clay stand so long. We weather the clay for some time, and let the first pugging stand for 24 hr. There is practically no loss in drying. The loss at the preheating kilns does not

exceed 3 per cent. My opinion is that in order to have longer life in the furnace, it is necessary to have some loss in drying and preheating; in other words, the greater the loss in drying and preheating, the longer the life will be of the retorts that stand up in the furnaces for a day or so.

The silica retorts do not bend at all, which is the principal reason for their longer life. Instead of having one large hole, as in the bottom of the middle of the nonsilica retort, small holes develop in different parts of the silica retort.

On straight Joplin concentrates, we are charging 57 lb. per cubic foot of retort capacity, not including the blue powder retorts, or 45.5 lb. including the blue powder retorts.

Our silica retorts do not have as long a life as given by Mr. Spencer. We would say that the cause of this is that artificial gas is not as hard on retorts as the natural gas, if it were not for the fact that one natural gas plant obtains a longer life than we do. We do not have a hydraulic retort press, which may account for the difference, or it may be caused by using unsintered ore.

The study and investigation of the causes of holes at certain places in the retorts is very interesting, but the investigation is slow, and so many elements influence results that it is a difficult matter to arrive at any definite conclusions. We do know, however, that the introduction of silica has increased the life of the retort.

W. R. INGALLS, New York, N. Y. (written discussion).—The distinctive features of what are being called silica retorts, or more specifically silica-flour retorts, are the use of very fine silica and a large proportion of it. These were introduced at Rose Lake plant by W. F. Rossman, who obtained a patent, and they are now adopted as standard practice in several zinc-distilling plants, especially those treating Joplin concentrates, which, of course, have a siliceous gangue. However, they have done very well in at least one plant where the ore that is treated is of more basic gangue.

Comparative tests that I have caused to be made have shown no superiority over the ordinary clay retorts in respect of heat conductivity. The silica-flour retorts certainly have a superior rigidity under heat and load in the furnace, and that is a quality to be prized. So many factors affect the life of retorts that comparisons are worthless unless they be made under identical conditions, beginning with the manufacture and curing of the retorts.

TABLE 4.—*Analysis of Materials Used at Donora Zinc Works*

Material	Ignition Loss, Per Cent	SiO_2, Per Cent	Al_2O_3, Per Cent	Fe_2O_3, Per Cent	CaO, Per Cent	MgO, Per Cent
Fireclay........	10.27	58.56	26.43	3.43	0.41	0.35
Grog..........	0.30	64.26	30.92	1.35	0.23	0.32
Silica..........	0.19	99.19	Al_2O_3 and Fe_2O_3 0.55		0.02	0.05

Density of charging is not to be associated too closely with the character of the retorts. Density, which means the pounds of ore charged per cubic foot, is governed by the weight per cubic foot of the ore itself, the proportion of reducing material that is mixed with it, and the manual or mechanical efficiency of throwing the mixture into the retort. At the present time the densest charging is done in the ordinary clay retorts. The silica-flour retorts are not, therefore, leading the way in that direction, but, of course, one kind of retort is capable of receiving as much charge as the other. Anyway, there is a limit to charging density for other reasons.

M. M. NEALE, Donora, Pa. (written discussion).—The materials used in the manufacture of high-silica retorts at Donora Zinc Works consist of fireclay from

the St. Louis district, grog composed of clean, evenly fired, broken saggers from the potteries at East Liverpool, Ohio, and silica ground through 140-mesh screen. The analyses are given in Table 4. The fireclay is weathered when received and usually is run direct through an American Clay Machinery Company's 9-ft. dry pan, elevated and screened through a vibrating screen set at a pitch of 55°. The screens are 3 mesh to the inch, 0.094-in. dia. wire. The materials passing through the screens go into a storage bin and the oversize returns to the dry pan. The grog is crushed and handled with the equipment used for the fireclay, except that a wire-cloth screen with a ⅛ by 3½-in. opening is used. The silica is received in bags and is hoisted and dumped into a bin on the level with the clay and grog bins. Screen analyses of the clay and grog are as follows:

Mesh	6	8	10	20	40	60	80	100	−100
Fireclay	1.0	5.6	6.0	49.5	20.8	9.2	3.2	0.8	3.9
Grog	9.4	28.1	14.0	37.6	4.9	1.4	0.6	0.2	3.8

In mixing, the materials are weighed in proportions of 250 lb. grog, 250 lb. silica and 500 lb. fireclay, into a dump bucket with a spring scale attachment, suspended on an overhead trolley. The materials in these proportions are then dumped into a hopper from which they are conveyed by means of a short screw conveyor into a cylindrical revolving dry mixer, discharging into an American Clay Machinery Co. combination wet mixer and pug mill, where water is added to give a moisture content of 12 per cent.

The ballots discharged from this machine are broken into lengths varying from 18 in. to 2 ft. and piled in the tempering room, where they are kept from five to seven days. Enough material is mixed and piled at one time to fill a dry room with retorts, 1140 retorts being the average capacity of each room. The mix as taken from the tempering room analyzes: ignition loss, 5.71 per cent; SiO_2, 67.66; Al_2O_3, 21.66; Fe_2O_3, 2.07; CaO, 0.22; MgO, 0.31.

After tempering, the material is run through an American Clay Machinery Co. pug mill, which produces a ballot 14 in. in diameter, which is cut by hand into lengths suitable for one retort. The retort is then made on a Wettengel hydraulic press with a pressure of about 2000 lb. per square inch.

Seven men are used to make retorts. Two pug-mill feeders, one pressman, one press operator and three retort truckers. The retorts are trucked and set in the dry room with a temperature of from 80° to 90° F. When the room is filled it is closed and left at room temperature for 15 days. The heat is then turned on and the temperature raised gradually. At the end of 45 to 50 days, when the retorts are needed, the temperature has reached 130° to 140° F. The dry room loss for the past five months has been 3.7 per cent.

The dry retorts are delivered to the kilns each morning. The kilns are filled and sealed: the fire is started and continued until time to change retorts the following morning. The kilns are fired by hand, with coal, and the temperature is raised gradually during the firing time. The kiln losses for the past three months have amounted to 0.7 per cent. The average life of retorts over the same period of time has been 41.4 days.

Failures of Cast-iron Kettles in Lead Refining

By Carl E. Swartz,* Maurer, N. J.

(New York Meeting, February, 1931)

For many years kettles used in the melting and refining of lead and other nonferrous metals and alloys have been made of cast iron. The logic of this probably lies in the fact that cast iron has been known as a material of construction for vessels of this type much longer than steel, and satisfactory fabrication from it has been more easily accomplished and better understood.

As the capacity of lead refineries has increased, it has been only natural to increase the size of kettles. Today kettles of 150 to 250 tons are not uncommon. As the size of the kettles increased and better shapes were designed, the uniformity of material of construction became more important. Records kept over comparatively long periods of time have shown that the length of service of kettles has not been uniform. Many causes contribute to this and the importance of each has not yet been evaluated. Kettle design, kettle setting, method of firing and material of construction have been changed from time to time to obtain better service, but the variation in length of service is still great. A kettle may fail during the first few heats or may last through more than one thousand heats.

The purpose of this article is not to present the conclusions of a finished piece of research on kettle manufacture and use, but to present a series of observations made over considerable time in the hope that others will publish data and that all may be correlated into a logical scheme which will result in better service and longer kettle life.

Possible Causes of Kettle Failure

The following outline suggests some possible reasons for the failure of kettles:

1. Improper heating.
   - *a.* Uneven heating over the surface of the kettle.
   - *b.* Rapid change in temperature.
   - *c.* Irregular use.
2. Faulty kettles.
   - *a.* Improper material.

* Research Department, American Smelting & Refining Co.

b. Improper design.

c. Insufficient or improper "heat treatment."

3. Chemical changes in kettles.

a. Transformation of combined carbon to graphite with its accompanying increase in volume and strains produced thereby.

b. Reactions between oxidizing gases, penetrating kettle structure, and graphite and iron producing iron oxide and causing an increase in volume.

4. Solubility or chemical reaction of some constituents of the kettle with the contents of the kettle and the gradual removal of the reacting constituent from the kettle structure.

Heating of Kettles.—Uneven heating of kettles is responsible for many kettle failures. This is shown by the fact that in some settings the failures of the kettles take place in the same location repeatedly. For example, kettles in some settings invariably fail on the side opposite the oil burner and near the flue outlet; others fail slightly above the point where the flame impinges on the kettle. When kettles are fired with coal or coke, they last over longer periods of time. Without doubt, this is due in part to the fact that the temperature is relatively uniform over the kettle surface and to the fact that temperature change is also comparatively slow. The fires are usually banked so that complete cooling is uncommon; and the less flexible coal fire and greater amount of brick work in the fire box act to decrease the rapidity of any temperature change.

If a coal-fired kettle is changed over to oil without completely redesigning the setting to use oil, kettle failures generally increase, due, in part at least, to the fact that the oil flame in a setting designed for coal firing produces uneven heating.

When a kettle is used continuously it is not subjected to extremes or rapid temperature changes; consequently, the best service comes from the kettles used regularly and continuously.

Faulty Kettles.—Of the various alloys of iron, cast iron is the most susceptible to mechanical and temperature shocks. It is therefore least suited to any use where such shocks are common. However, the manufacture of cast-iron kettles at present is cheaper than the manufacture of kettles from other materials, which, no doubt, accounts for their wide use.

In designing a kettle to resist mechanical shocks incident to desilverizing and drossing practice, it is necessary to use a cross-section of considerable size. Unfortunately this thickness is sufficient to produce a temperature gradient through the metal. This temperature gradient then brings about a difference in expansion, with its accompanying strains making for compression on one side and tension on the other. This temperature difference is augmented by the poor conductivity of the

graphite, which amounts to at least 10 per cent. of the volume of the average casting. Hence the kettle should be as thin as is consistent with strength, in order to minimize loss in heating efficiency and decrease expansion differences.

Manufacturing and Service Practice.—The best kettle settings are designed to produce an even heat over the entire surface of the kettle. The location and size of the burner, the shape and length of the flame, and the velocity and path of the combustion gases are important considerations.

In manufacturing a kettle, it is generally recognized as essential that the mix be of high-grade pig iron with little or no scrap, and that care be used in casting to produce a kettle of uniform thickness, free from deep sand marks. Very slow and even cooling minimizes casting strains. In the last few years some special pig irons, such as Mayari iron (containing chromium and nickel), have been used, but superior results have not been definitely proved.

Some plants have a so-called annealing process. When a new kettle is set, it is filled with water and boiled dry. During this treatment the temperature attained is 212° F. Casting stresses are relieved at elevated temperatures up to 1050° or 1100° F. Therefore this practice is of doubtful value.

Another common practice is the turning of a kettle through 45° in its setting once a month. This aids in equalizing any tendencies toward local heating and the strains and chemical reactions taking place as a result of local overheating.

Chemical Changes in Kettle.—It has long been known that when cast iron is repeatedly heated and cooled, there is an increase in volume. According to Remmers,[1] this is due to alternate precipitation and solution of graphite, followed by oxidation of the matrix material after the graphite has been burned out. This volume change results in strains in the metal which cause cracking at that point. With uneven or rapid temperature change this tendency to crack is augmented. No evidence has been found to date to show the importance of these phenomena in kettle life.

Solubility or Chemical Reaction of Some Kettle Constituent with the Bullion.—The failure of lead kettles might be due to the solubility or chemical reaction of some constituent of the iron alloy on the bullion. As indicated in Fig. 4, the lead bullion seems to follow the graphite. However, in cracking, the path of the crack would follow the graphite and lead would follow the crack. If there were any tendency toward solubility or reaction between the kettle and the bullion, evidence would

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[1] W. E. Remmers: Permanent Growth of Gray Cast Iron. *Trans.* A. I. M. E., Iron and Steel Div. (1931) 219.

be found in the microstructure of the inside surface of the kettle. Such evidence has not been found. Examination did show minute cracks along the inside surface where lead had penetrated ¼ to ½ in. As will be discussed later, small cracks have been found inside the casting, which did not come to the surface.

Therefore the main causes of kettle failure are to be found in the first three divisions of the outline; namely, improper heating, faulty kettles and chemical changes in the kettles. Improved design of kettle settings with better shape and cross-section of the kettles has increased the life of kettles substantially.

Microstructure of Kettles

The best type of microstructure for refining kettles is not precisely known. Certain general conditions have been found in cases of early failure, but sufficient work has not been done to indicate the relative importance of various factors. In many cases where early failure has been noted a coarse black fracture has been found, rather than the fine even gray fracture which is preferred and is striven for by founders. The importance or unimportance of type of microstructure or its alteration has not yet been evaluated.

Table 1 gives a summary of data on the various kettles discussed.

Table 1.—*Data on Kettles Discussed*

Kettle No.	1	2	3	4	5
Capacity, tons	165	60	110	165	70
Fuel	Oil	Oil	Oil	Oil	Oil
Use	Molding 40 charges dezincing 7 charges premelting 25 charges	Drossing	Desilverizing	Premelting	Desilverizing
Life charges	72	381	190	11	1180
Analysis, per cent.					
Combined C		0.52	0.63		Total C, 3.45
Graphite		2.80	2.30		
S		0.09	0.12		0.078
Mn		0.79	0.79		0.63
Si		1.31	1.26		1.23
P		0.09	0.245		0.09
Rockwell hardness					
In rim	B 60	B 60	B 77	B 64	
At failure	B 37	B 44	B 62	B 46	B 57

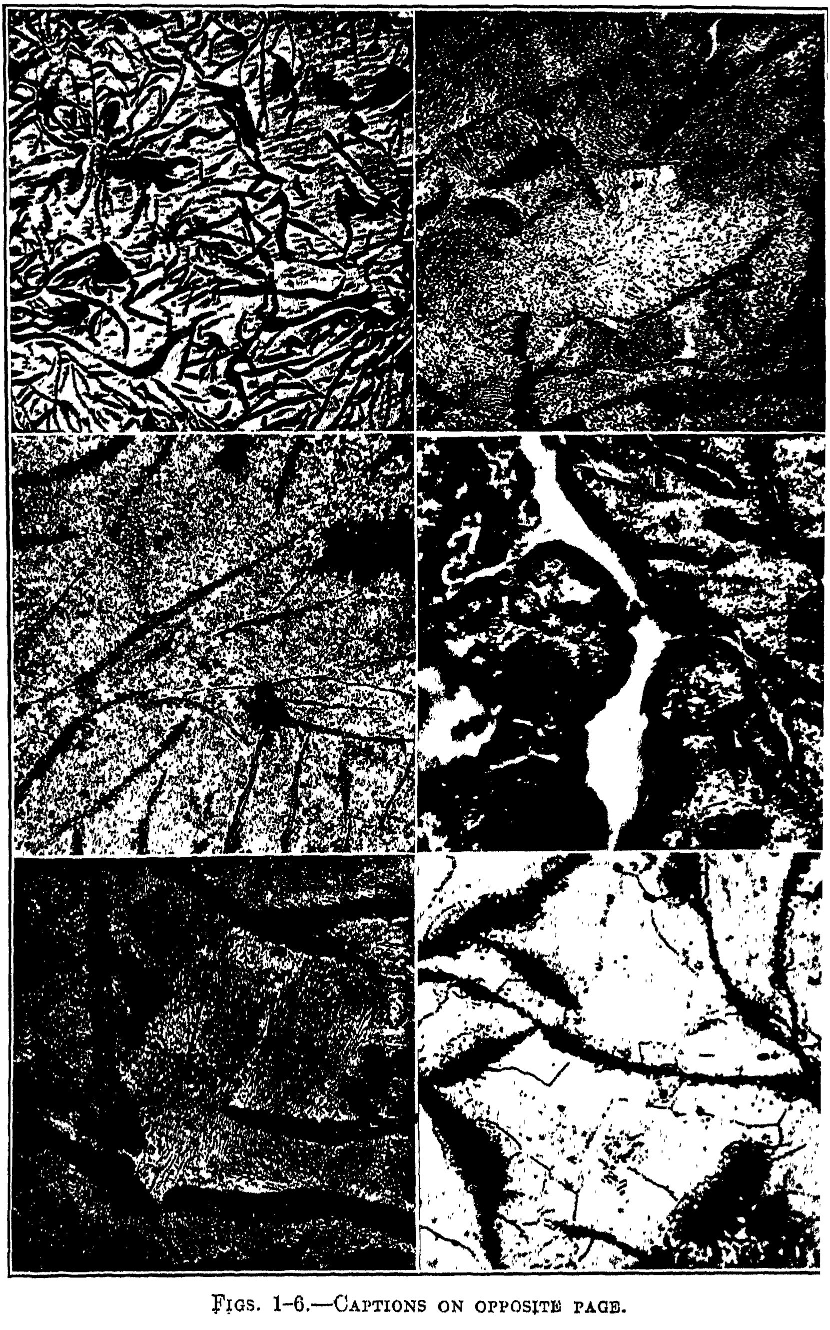
A survey of various manufacturers showed that kettles were cast from metal made in a cupola without the use of scrap. Molds were made of loam and used in the upright position. After casting, the assembly was undisturbed for from two to seven days.

In studying the microstructure of the kettles, specimens were taken from two or three parts of each casting. A rim specimen was examined to find the structure of the kettle at the time of placing in service. It was assumed in this case that the rim did not reach a temperature high enough to alter the original structure. A specimen was taken from the kettle at or near the point of failure. Often a specimen was taken from a point on the kettle opposite the failure, to determine any difference in structure between the point of failure and other parts of the kettle below the lead line.

In the examination of the various specimens, attention was given to size, shape and number of graphite particles present; to the presence and amount of temper carbon; to alteration of structure from heat; and to any cracks present in the microstructure.

A typical failure is illustrated in kettle 1. This molding kettle started to leak shortly after setting and continued until charge 72, when it was removed from service. The fracture of this casting was very dark, indicating that the break occurred largely through graphite rather than iron and graphite. Evidence was found of penetration of liquids as far as the center of the casting. The microstructure indicated a porous open structure. The graphite particles were long and narrow and connected with one another in many cases. Fig. 1 shows this structure in the unetched condition. Fig. 2 illustrates the type of structure found in the rim of kettle 1, the usual structure of gray cast iron, graphite-pearlite structure. Fig. 3 shows the structure found in this kettle at the fracture. Two changes are notable here: (1) The pearlite has changed. At the rim we found the normal ferrite and cementite laminations of pearlite, whereas here we find very little normal structure. An alteration is in progress whereby the cementite is gradually agglomerating. (2) In many parts of the microstructure, we find minute cracks in the matrix. These generally originate at some of the smaller graphite particles and continue a short distance or progress to a near-by graphite particle. In the etched condition it is sometimes difficult to distinguish between fine graphite particles and the cracks, but in the unetched condition the difference is readily apparent. (Compare Fig. 7.) In Fig. 3, three cracks originate at the graphite particle in the lower left of the photomicrograph.

Kettle 2 was used for drossing and gave the average length of service. Fig. 4 shows a cross-section of a crack which ended the service of this kettle. The large light areas within the graphite particles are lead, which has followed the crack as it progressed from one graphite particle to the next. Whether these cracks are the same as those found in the interior of the casting has not been established. It is quite probable that they are not, as no cracks have been found in the structure larger than those shown in the accompanying photomicrographs. It is probable



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Figs. 1–6.—Captions on opposite page.

that as the graphite particles increase in size, the pressure exerted causes these minute fractures which do not increase in size to any extent after the initial pressure is relieved. As will be shown later, these cracks have been found in sound parts of the kettle at points remote from the failure.

The rim of this kettle is composed of the usual gray cast iron, largely graphite, pearlite and ferrite. Fig. 5 shows a section of the rim in the etched condition revealing these constituents in the normal form. Fig. 6 illustrates the structure of the kettle below the lead line and away from the fracture. The change in pearlite is very apparent. The pearlite has broken down and the ferrite and cementite have been divorced from one another. This indicates that the kettle has been heated to temperatures of at least 1100° F. The dark irregular line in the center of the field is a rather long crack connecting two masses of graphite. It is evident from this photomicrograph and Fig. 8 that a further change has taken place. The spheroidized cementite does not appear to be present in quantity sufficient to account for the cementite originally present. The structure resembles to some extent the structure observed in cast iron that has been malleableized. In other words, the pearlite has altered by divorcing and spheroidization of the cementite. The cementite has further decomposed to form graphite and part of the graphite has been burned out by the oxidizing gases present in the combustion gases.

Fig. 7 shows a field from the same specimen as Fig. 6, in the unetched condition. The fine cracks are much more apparent here.

Fig. 8 illustrates the structure near the fracture. This is essentially the same as Fig. 6 except that there appears to be slightly more temper carbon present. No microscopic cracks could be found in this specimen. The fact that the microstructure of the last two specimens is essentially of the same type indicates that the kettle was heated rather evenly.

There is, however, some difference in the structure of the two sections of the kettle. The structure away from the fracture, Fig. 6, is more uniform. The alteration in the pearlite progresses more evenly from inside to outside. Only occasional and isolated spots of undecomposed eutectic are to be found. At the fracture, Fig. 8, there is more temper carbon. The structure is more spotty than in the sound part of the kettle. Near the inside of the kettle pearlite is unaltered. There is rather an abrupt change from unaltered to altered pearlite about one-third the distance between the inside and outside surfaces of the casting.

FIG. 1.—KETTLE 1. NEAR FAILURE. UNETCHED. × 25.

FIG. 2.—KETTLE 1. RIM STRUCTURE. ETCHED IN ALCOHOLIC PICRIC ACID. × 250.

FIG. 3.—KETTLE 1. STRUCTURE AT FRACTURE. ETCHED IN ALCOHOLIC PICRIC ACID. × 500.

FIG. 4.—KETTLE 2. MINOR CRACK AT FAILURE. UNETCHED. × 100.
Light areas are lead penetrating in crack in graphite masses.

FIG. 5.—KETTLE 2. RIM ETCHED IN ALCOHOLIC PICRIC ACID. × 250.

FIG. 6.—KETTLE 2. SOUND PORTION OF BODY OF KETTLE. ETCHED IN ALCOHOLIC PICRIC ACID. × 250.

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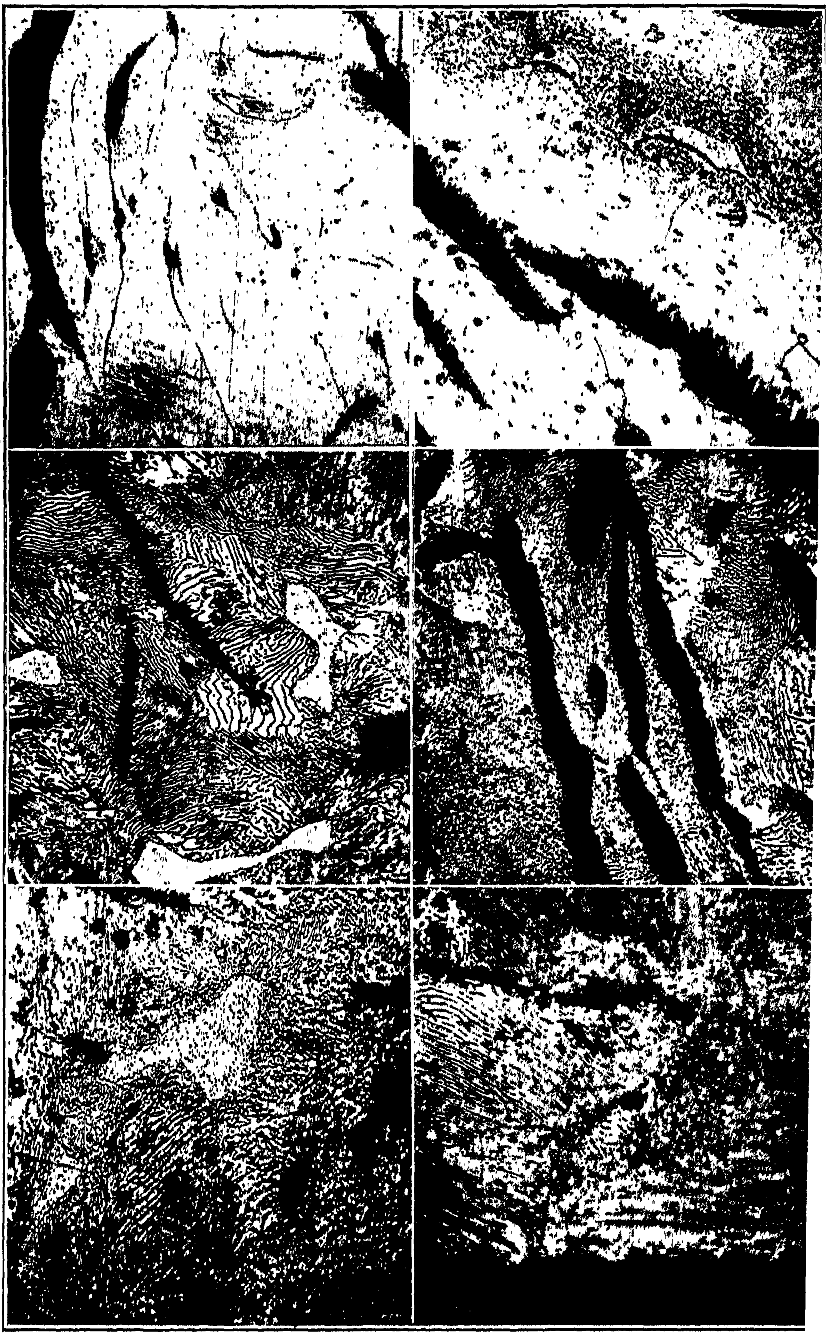

FIGS. 7–12.—CAPTIONS ON OPPOSITE PAGE.

The photomicrographs were taken near the center of the cross-section of the casting. A band was found along the inside of the kettle, which had the normal graphite-pearlite structure. The change from this to the divorced structure was abrupt. The cooling effect of the lead probably kept this inner layer of the kettle at a temperature too low for spheroidization to take place. The graphite inclusions at the outside of the kettle were ragged at the edges, indicating oxidation at the interface. This condition was noticeable also at the cracks but in both cases there was a rapid change to clear-cut graphite particles as the interior of the section was reached.

Kettle 3, used for desilverizing, illustrates the case of a kettle of which the life was shorter than usual, and in which there was evidence of uneven heating. Fig. 9 represents the structure found in the rim of kettle 3, the usual pearlite-ferrite structure. Fig. 10 shows the structure of the kettle in a sound part, below the lead line and is essentially similar to Fig. 9. Fig. 11 represents the structure of the kettle at the fracture. Here we find more temper carbon than previously and the process of divorcing of the pearlite well under way but not as complete as in Figs. 6 and 8. The light grains in these three figures are phosphide eutectic. This is the only kettle in which this constituent has been noted. While the high phosphorus content of the metal was noted, it is probable that it had no detrimental effect on the kettle life. No evidence of fusion of the eutectic was noted.

The pearlite in the outside three-quarters of the kettle cross-section is more completely decomposed than in the inside one-quarter. An interesting observation was made in this connection. Where the specimen was allowed to stand in the laboratory for a few minutes after polishing rust was formed on the polished surface, starting from the inside edge and working toward the original surface of the kettle. The rusting action ceased, however, at the line corresponding fairly well to the division between slightly decomposed and more nearly completely decomposed pearlite. Thus the more normal structure corroded while the altered remained bright.

The difference in structure between the metal at the fracture and that in the sound portion of the kettle indicates that there was a considerably higher temperature at the fracture than at other points in the body of the kettle.

FIG. 7.—KETTLE 2. SOUND PORTION OF BODY OF KETTLE. UNETCHED. × 100.
FIG. 8.—KETTLE 2. STRUCTURE NEAR FRACTURE. ETCHED IN ALCOHOLIC PICRIC ACID. × 250.
FIG. 9.—KETTLE 3. RIM. ETCHED IN ALCOHOLIC PICRIC ACID. × 250.
FIG. 10.—KETTLE 3. SOUND PORTION OF BODY OF KETTLE. ETCHED IN ALCOHOLIC PICRIC ACID. × 250.
FIG. 11.—KETTLE 3. NEAR FRACTURE. ETCHED IN ALCOHOLIC PICRIC ACID. × 250.
FIG. 12.—KETTLE 4. RIM. ETCHED IN ALCOHOLIC PICRIC ACID. × 500.

Kettle 4, used as a melting kettle, illustrates the type of early failure thought to be produced by excessive heating and coarse metal mix. Fig. 12 is representative of the structure of the rim of the kettle. Figs. 13 and 14 show two structures found on the same specimen. In the latter the cementite has agglomerated into spheres somewhat larger than those in Fig. 12. Since this kettle lasted only 11 charges, it is possible that the damage was done by a single overheating.

13 14

15 16

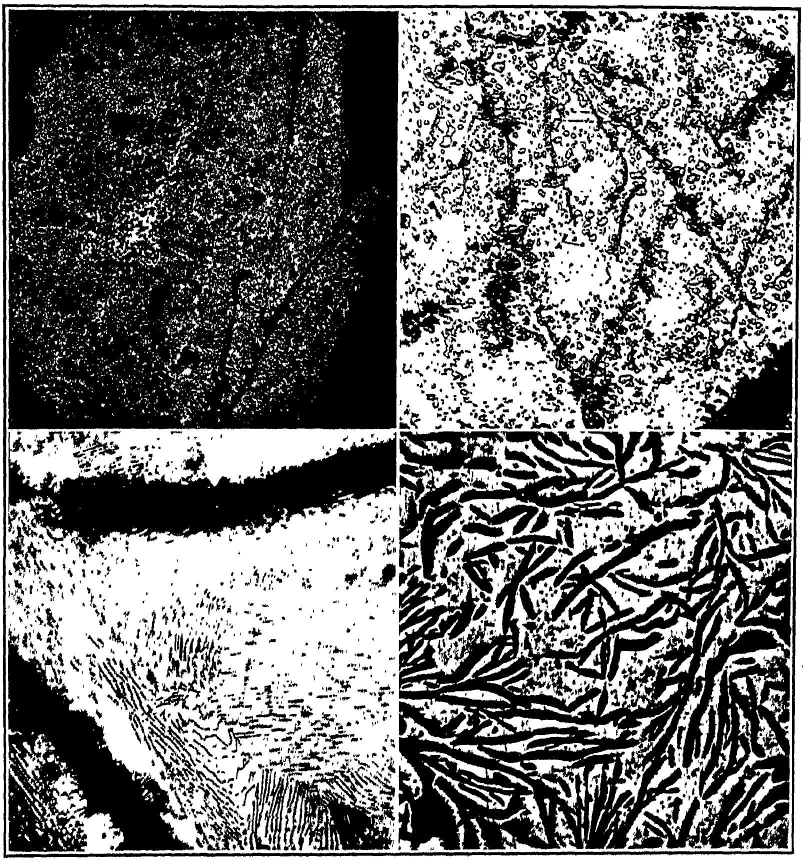

FIG. 13.—KETTLE 4. NEAR FRACTURE. ETCHED IN ALCOHOLIC PICRIC ACID. × 500.
FIG. 14.—KETTLE 4. NEAR FAILURE. ETCHED IN ALCOHOLIC PICRIC ACID. × 250.
FIG. 15.—KETTLE 5. SOUND PORTION OF BODY OF KETTLE. ETCHED IN ALCOHOLIC PICRIC ACID. × 250.
FIG. 16.—KETTLE 5. SOUND PORTION OF BODY OF KETTLE. UNETCHED. × 25.

The microstructure of this kettle was much coarser than usual, being porous, open and granular in appearance. As in the previous kettle, rusting took place with ease in a small band along the inner side of the kettle.

Kettle 5 illustrates the best type of service of which we have record. This kettle gave service of nearly 1200 charges. Unfortunately specimens were not obtained of the rim and point of failure but only from a sound portion of the kettle below the lead line, so one can but guess at the type of structure in the rim and at the point of failure. However, Rockwell hardness tests made over the surface of the specimen submitted showed more uniformity than other specimens tested. When the kettle was broken to pieces for scrapping, it took more drops of the ball than usual, showing greater strength. The fracture was very gray, indicating that breakage was taking place through metal to a greater extent than in the other specimens examined. The evenness of the break indicated absence of hard spots. Fig. 15 shows the structure of the specimen. After nearly 1200 charges, we still have a normal structure with little temper carbon. The structure changed very little through the thickness of the casting. Fig. 16 shows the unetched specimen at low magnification. Although the graphite particles are somewhat longer than usual, they seem to be separated from each other by considerable metal in many cases, with no evidence of oxidation at the interface.

Summary

From the cases cited, which are typical of those studied, it is not possible to formulate definite conclusions, except of a very general nature. Iron which on fracturing has shown the coarse open dark gray structure has given much shorter service than the iron which showed the fine light gray fracture. In the majority of kettles examined, the normal matrix structure as shown by the rim has undergone some change in the kettle body during the kettle service. The laminated pearlite has divorced more or less completely to ferrite matrix in which cementite is imbedded. These cementite particles have in some cases become almost spheroidal.

In many cases fine cracks were observed in the microstructure of the heated sections of the castings. These originated in a graphite particle and continued a short distance or connected with another graphite particle.

It is probable that one or more of these phenomena are responsible for the failure of the kettles. Which ones are responsible and to what extent are questions that further investigation will decide.

DISCUSSION

(Carle R. Hayward presiding)

J. O. Betterton and C. W. Hanson, Omaha, Neb. (written discussion).—Dr. Swartz' paper brings out a great deal of new and interesting information relative to the microstructural and chemical changes occurring in cast-iron kettles in use and subjected to usual heating conditions met with in lead-refining practice. It should

prove of great value when considered with other important factors in analyzing the causes of such failures.

The purpose of our comments is to supplement the subject matter dealt with in the paper by an addition of knowledge gained through experience records of kettle breakage at several refineries over a period of years.

In present lead-refining operations kettle units of 200 to 250 tons capacity have been successfully developed without decreasing, and in fact increasing, the average life of kettles. This probably will prove the maximum desirable, as inability to transport larger kettles, together with questionable advantages in operation, will no doubt place this limit on size, independent of the question of the life of larger kettles.

As the shape of a kettle is an important consideration in evenly distributing stresses due to contraction and expansion, it has evidently been the opinion of most lead refinery metallurgists that kettles of true hemispherical section, or hemispherical kettles with straight sides at the top, are best for the large sizes, as such kettles are in almost universal use. Accordingly, we will confine our comments to the use of such kettles.

It has been our experience with large round kettles, as well as with the smaller sizes, that failures occur by the abrupt formation of a crack or cracks from 1 to 3 ft. long in the sides or the bottom of the kettle. Only in rare instances do cracks develop in the rim and lengthen downwards. The location of the cracks is variable and cannot be predicted for any particular kettle setting with reference to burner or flue, provided obviously defective firing arrangements are not used. Apparently they are caused by accumulated stresses set up in the structure as a whole, which attain great magnitude, and which overcome the resistance of the metal along the line followed by the crack. While, of course, there are instances of breakage being due to a small crack widening or lengthening with continued use of the kettle, we believe that the important ones to guard against are those that occur abruptly and disrupt the metal by the formation of long cracks and render it practically useless for further service.

Discussing the outline of the probable causes of these failures as set forth in Dr. Swartz' paper, as follows:

1. *Improper Heating.* (*a*) *Uneven Heating over the Surface of the Kettle.*—There is no doubt about the importance of this on kettle life, and much improvement has been made in the past five or six years in the design of proper kettle settings, which has resulted in increased kettle life. Large kettles now in use, in all instances that we know of, provide sufficient combustion space beneath the kettle, and also have the burner set low enough to properly provide for the reduction of velocity of the combustion gases and to prevent flame impingement on the kettle itself. In many instances this is probably considerably benefited by short combustion chambers attached to the kettle, to reduce the velocity and spread the hot gases before they contact with the kettle itself.

Many efforts have been made to circulate the combustion gases within the setting so as to get a long path of travel in contact with the kettle before the gases reach the flue. While this point has possibilities for fuel economy, it is questionable whether it improves kettle life, as the added velocity given the combustion gases tends to carry a cutting flame against the kettle. Since the adoption of oil as fuel on most refining kettles, this idea has persisted, but after considerable experimentation with a number of these multi-pass settings, we believe that an open setting allowing free travel of gases to an outer flue is the most practical and desirable arrangement.

(*b*) *Rapid Change in Temperature.*—This is probably a cause of kettle failure not fully appreciated in the past, especially in the rapid heating of a kettle. In most instances the temperature of the metal is being observed in the kettle and the operation controlled thereby, while the important question, of course, is the maximum temperature to which the shell of the kettle is being subjected. There is little doubt that

with rapid reheating these temperatures are much higher than would be anticipated, and result not alone in deterioration of the metal through crystalline and chemical changes but also subject the kettle as a whole to structural stresses of great magnitude; at times, more than the strength of the metal will stand.

(c) *Irregular Use.*—There is little doubt that the optimum condition for lengthening the life of cast-iron kettles is the constant use of the kettle through a comparatively small range of temperature. Where kettles are used irregularly, especially for a number of purposes, our records indicate unsatisfactory life.

Kettles often are changed in the setting and before failure, by shifting the kettle about 45° at reasonable intervals, in order to move the parts directly in contact with the maximum heat to cooler portions of the setting. While this practice seems reasonable, we have obtained much evidence in our experience that it is not beneficial. The rim of the kettle usually is somewhat uneven, and the kettle seldom rests uniformly on the mantle plates of the setting; in fact, sometimes it is necessary to place shims on the setting to give proper bearing to the portions which do not rest evenly. After a kettle has been in use in a setting for a time, it gradually adjusts itself to the position, and if removed, turned, and replaced, it must again adjust itself, which means that an entirely new set of strains is set up, as there is usually distortion in the casting. On the whole, we believe this practice will increase breakage and that once a kettle is placed in a setting, the best practice is to let it remain in the original position until the life of the kettle has been obtained.

The practice of boiling to dryness water containing lime in a new kettle before use goes by the misnomer "annealing." This practice or some modification of it is persistent in most refineries. A double purpose is sought: first, the preparation of the inner surface of the kettle by means of a lime wash, and second, the relieving of strains in the casting which might cause breakage on a first heating.

While a temperature of 212° F. is far below that at which crystalline changes occur in the metal, and the effect thus very problematical, many kettles are moved in from open yards during cold weather, often in a wet or icy condition, so that putting them into service in this way does offer some assurance of proper slow heating up of the kettle.

2. *Faulty Kettles.* (a) *Improper Material.*—It has been proved conclusively that a high grade of pig iron must be used to insure long kettle life, therefore it has become most uncommon for large foundries to use scrap iron in these castings. In recent years tests on a fairly large scale have been conducted to determine the value of nickel and chromium in cast-iron kettles, these constituents being added by means of a definite proportion of Mayari iron in the mix. Like most results, they are not concordant, and as yet insufficient, as these tests require a long time, but on the whole there is considerable indication that the practice is an improvement.

At one plant careful analyses have been made over a period of five years in an effort to draw definite conclusions as to amounts of carbon, phosphorus, silicon, sulfur, manganese, chromium and nickel which are desirable in kettle castings. Probably because of the many causes of kettle failure, it has not been possible to draw any definite conclusion from these records as to the proper composition for the metal. Both good and inferior results have been obtained in cases where carbon was as low as 2 per cent and as high as 3.4 per cent, with silicon as low as 0.8 per cent and as high as 1.5 per cent, with manganese as low as 0.6 per cent and as high as 1. 4 per cent. There has been little variation in the content of sulfur and phosphorus in the various kettles tested and it is the general opinion that neither of these elements should much exceed 0.15 per cent.

(b) *Improper Design.*—The proper thickness of a kettle casting has been much discussed. The consensus of opinion has been that the kettle should have a relatively thick bottom, with the section of metal gradually decreasing toward the rim. Recent tests with kettles of uniform thickness and others with only very small increase of

bottom thickness over that at the rim, not to exceed ½ in., appear to disprove this assumption.

However, with a marked increase in the size of cast-iron kettles, it is probably desirable to increase the thickness of cross-section almost proportionately. In order to withstand mechanical shocks and also the considerable distortion occurring in the kettle in use, we believe this is essential. On the other hand, excessive thickness only adds unnecessarily to the cost and decreases the thermal efficiency of the kettle in service.

Uniform cross-section of metal is desirable, and our experience with kettles not manufactured true to the drawing, and as a consequence appreciably thicker on one side than the other, has resulted in sufficient failures to convince us of the importance of this matter.

In many instances it is either necessary or convenient to equip a kettle with a hollow spout in order to tap the lead from the kettle. These spouts are cored bosses integral with the kettle and in order to give sufficient thickness to the walls, it is necessary to make this and the adjacent area thicker than the main portion of the kettles. Records over a sufficiently long period have shown conclusively that such kettles fail prematurely and that the failure in a majority of the cases occurs either in the spout itself, directly back of it, or on either side. This is a good illustration of the effect of unequal cross-sectional area setting up excessive local strains which result in failure.

(*c*) *Insufficient or Improper Heat Treatment.*—A close study of foundry practice may well result in developing important information bearing upon the life of cast-iron kettles.

(*d*) *Chemical Changes in Kettles.*—Reactions with oxidizing gases and the transformation of combined carbon to graphite both result in an increase in volume and the production of additional strain. As the photomicrographs clearly show, these cause small cracks to start from the graphite inclusions. In short, both actions probably tend to a deterioration and weakening of the cross-section of the kettle and while this has not been definitely shown by tests, we believe it could be properly brought out by making tests of tensile strength on samples taken at the rim and at a point closely adjacent to the point of failure.

To sum up, a careful consideration of kettle records intended for the purpose, over a number of years, and for service under a large variety of conditions, convinces us that there are so many factors involved in kettle failures that it is hardly possible to correctly assess any one of them, except in a general way. Cast iron at best is unreliable as regards uniformity of strength, even when a kettle is properly manufactured to drawing, and there is no doubt in our minds that kettles of the identical size and shape, and produced by a good manufacturer during the same period, would show, if it were possible to properly test them, a marked nonuniformity of strength of cross-section in different parts of the kettle. In brief, we believe that the maximum strains that kettles will stand without rupture are variable as they come from the foundry and that there may be early failures even when the use of the kettle is not severe.

There are few refinery men who have not known occasionally kettles that gave remarkable life, even under conditions of poor firing arrangements and known excessive temperatures at times. On the other hand, there is no question about the value of many recent developments in better kettle settings and design of shape and cross-section of the kettle; also of the value of closely controlled operating conditions, especially in regard to temperatures and rapidity of heating of kettles. Mr. Swartz, in bringing out crystalline and chemical changes accompanying uneven heating of kettles and the detrimental effects of temperatures beyond certain limits, adds another new and valuable viewpoint for a consideration of the whole problem.

G. E. Johnson, East Chicago, Ind. (written discussion).—Mr. Swartz is to be congratulated on initiating a discussion on the life of cast-iron kettles. There is a vast accumulated experience in the use of kettles in lead smelters and refineries. It is hoped that this information will become available for general discussion either by presentation of individual papers or discussions of this and subsequent papers on this subject.

We agree with Mr. Swartz that the three main causes of kettle failures are improper heating, faulty kettles and chemical changes in the kettles.

Several years ago we made a rather extended investigation into the possible chemical changes in kettles in service and the effect of these chemical changes upon the kettle life. Observations were made upon the conditions and the service given by 14 kettles, including chemical analyses and micrographs of the kettle material taken from the rim and point of failure. In every case, a change in the structure and chemical composition of the iron was found to have occurred at the point of failure. The pearlite had changed either to a "spheroidized cementite" or into ferrite and graphite. This change was one which could have been caused by overheating, and represented a weakened condition of the iron. We concluded, therefore, that the primary cause of the failure was local overheating. Our investigations of the changes in the structure of the cast iron at the point of failure definitely confirm the results reported by Mr. Swartz.

We recognize that the kind of service required of a kettle will lead to a gradual deterioration of the cast iron and an ultimate failure from the same causes and of the same character as that noted in our investigation. The advantage to be gained by such an investigation is in the possibility of reducing the rate of deterioration of the cast iron.

The kettles represented in this investigation were all coal-fired and were used for a variety of services. The kettles subjected to the most severe service had the shortest life. As a result of these tests, our kettle settings were changed to reduce the possibility of local overheating. Two additional new settings were constructed in 1929 and designed to eliminate local overheating.

The practice of turning a kettle in its setting undoubtedly distributes the strains due to local overheating. We abandoned this practice some time ago, on account of an increase in our kettle failures. We concluded that the readjustment of the kettle in the setting after turning developed new strains which were more serious than those due to local overheating.

It is well recognized that improvement in kettle life is slow, because of the large number of factors involved and the long period over which tests must be made to draw any definite conclusions. From the information given in Table 1, the maximum life obtained was 1180 charges from kettle 5. The capacity of this kettle was 70 tons. The longest period of service was obtained from kettles of 60 and 70 tons capacity, indicating that the life is considerably reduced as the capacity of the kettle increases. This confirms our observations.

It is a question whether foundry practice has kept pace with the increase in capacity of kettles specified by lead refineries. Have the possibilities of adding beneficial metals to cast iron to reduce the rate of deterioration of the metal been exhausted? Have the users of kettles fully realized the limitations of cast iron for this service and sufficiently improved the kettle design and settings corresponding to the increase in capacity desired? These questions suggest a symposium on kettles in conjunction with the American Foundrymen's Association.

Our experience with cast steel kettles has been rather limited, but decidedly unsatisfactory. In one particular case in which the foundry cast the kettle "bottom side up," the kettle failed after three charges, because of a porous structure at the point where the "riser" was located. It has been suggested that cast steel kettles

be made from steel with a manganese content and general analysis approximately that used for steel ingot molds in steel plants. We have not tried steel kettles of this composition.

At the time we were investigating cast steel kettles, we assumed that steel kettles could be repaired by welding. It is our opinion that a satisfactory repair of cast steel kettles by welding is impractical, and that the assumed advantage is largely imaginary.

Some method of definite detection of imperfect or porous castings which could be used by the foundry prior to shipment would be of considerable advantage. A development of some method similar to the use of the X-ray for detection of unsound castings used by oil refineries might solve this problem.

Any comprehensive study of factors affecting kettle life should consider the variation of working conditions between melting, drossing, desilverizing and molding kettle service. Some satisfactory definition of the term "charge" would help clarify future discussions. Any method of approximately equating the difference in working conditions in the definition of the term "charge" would be a decided benefit.

J. J. MULLIGAN, East Chicago, Ind. (written discussion).—I agree with Mr. Johnson that a study of the factors affecting kettle life should take into consideration the various uses to which the kettles are put, and also the design of the fire box, type of fuel used, the period during which the kettle is under fire, maximum and minimum temperatures, etc.

My experience has been principally with cast steel kettles. The kettles that we are now using have a capacity of 130 tons of lead, have an inside diameter of 11 ft. 8 in., are 5 ft. 9 in. deep with a 6-in. rim. The shape is semispherical and the kettle has a thickness of approximately 4 in. at the bottom and tapers to 2.5 in. at the rim. The chemical analysis of the steel used in our kettles has been approximately as follows: carbon 0.29 per cent, manganese 0.69 per cent, phosphorus 0.035 per cent, sulfur 0.041. These kettles are used for remelting purposes, starting with a charge of cold lead material.

In one operation where these kettles are used, melts are made only three or four times a week with a day or two or even more intervening between the melts. No fire is maintained while the kettle is not in operation, so that the kettle and the setting become fairly cool between melts. These kettles are under fire for a period of about 20 hr., starting with the charging of the kettle and covering the last few hours of the period when the metal is held at a molding temperature. The maximum temperature of the molten metal is 900° F. The kettles in this operation, towards the end of their life, have a tendency to come up in the bottom as though the metal were pushed up from beneath, and finally show cracks of various lengths on the inside of the bottom.

The other remelting operation is similar to the one just described except that the maximum temperature is 1000° F. However, the kettles in this operation stretch in time, beginning with a point about one-third the way up from the bottom, and in some cases the metal has been drawn down to a thickness of 1 in., having started perhaps with a thickness of 2½ to 4 in. This stretching of the metal seems to exaggerate the porosity, which originally was present in certain spots in the kettle, and leaks develop at these porous spots.

Formerly we gave our kettles a quarter turn about every month or so, but more recently have come to the conclusion that turning the kettles is not as good a practice as to allow them to stay in the position in which they were originally placed. We have not had enough experience, however, to say that the nonturning of cast steel kettles is beneficial.

For a number of years our kettles were coal-fired and we had excellent results; a period of bad results occurred during 1917 to 1919, but beginning with 1920 results again became satisfactory. At about that time we changed over to oil firing,

but have noted no change in the life of the kettles. At the time that the change was made in the method of firing there were two kettles in service that had had normal life, and these two kettles under oil firing ran to more than double normal life, which would seem to indicate that these were two particularly well cast kettles, and the method of firing in this case was not an important factor. At present we are using oil firing with a Dutch oven fire box to prevent the impinging of the flame on the kettle.

From our experience, it seems that failures in cast steel kettles more often result from faulty casting than from other reasons. Occasionally we have received kettles containing cracks, which if put in service have developed leaks through such cracks. Of course improper heating and chemical changes may accelerate the failure of the kettle, but in many cases we have found porous spots on new kettles; we marked these porous spots before putting the kettles in service and in almost every case the leakage on kettles so marked has occurred at these faulty spots. It has also been our experience that the kettles that leaked during the first few charges failed completely within a comparatively short time, whereas with kettles that developed no leaks in, say, the first hundred charges, there was a strong probability of getting normal life.

D. F. McFarland, State College, Pa.—I am convinced that enlargement of size in use is the principal cause of failure. This is a common phenomenon known as "growth" of cast iron. Experiments which have been made at State College indicate that this may be remedied by use of nickel and chromium.

Investigation of Anodes for Production of Electrolytic Zinc, II

By H. R. Hanley,* C. Y. Clayton† and D. F. Walsh,‡ Rolla, Mo.

(New York Meeting, February, 1931)

The characteristics of alloyed anodes and their influence on the products of electrolysis and power consumption have been noted previously in the literature.[1] This paper presents data in continuation of the subject given in an earlier paper by the authors.[2]

As noted in the authors' previous paper, the objectives of this work are the reduction in the cell voltage and the purity of the cathode product resulting from the use of alloyed lead anodes.

The fundamental facts developed in the previous paper were: (1) that the presence of a small amount of calcium lowers the anode potential approximately 50 per cent. below that of pure lead; and (2) that thallium when present to the extent of 4 per cent. rendered the anode practically stable, thereby lowering the contamination of the cathode by lead to a very insignificant amount.

The specific features determined in this recent work are as follows:

1. The lowest limit of calcium in the lead which will still give maximum lowering of the anode potential.
2. The characteristics of the thallium-calcium-lead anode.
3. The characteristics of the silver-calcium-lead anode.
4. The characteristics of the thallium-silver-calcium-lead anode.

It was determined that 0.1 per cent. calcium in the anode was sufficient to lower the anode potential approximately 50 per cent. Table 1 shows the characteristics of alloyed anodes tested. The anode composed of 0.1 per cent. calcium, 4 per cent. thallium, and 95.9 per cent.

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[1] Among others, see C. G. Fink and Li Chi Pan: Insoluble Anodes for the Electrolysis of Brine [*Trans.* Amer. Electrochem. Soc., (1926) **49**, 83-134] and other papers by these authors.

U. C. Tainton, A. G. Taylor and H. P. Ehrlinger: Lead Alloys for Anodes in Electrolytic Production of Zinc of High Purity. *Trans.* A. I. M. E. (1929) 192–200.

[2] H. R. Hanley, C. Y. Clayton, D. F. Walsh: Investigation of Anodes for Production of Electrolytic Zinc. *Trans.* A. I. M. E. (1930) 275

lead showed the greatest stability, and therefore produced zinc cathodes with the lowest lead content. The lowering of the anode potential in this case was approximately 40 per cent. The influence of thallium partly neutralizes the specific effect of calcium in lowering the anode potential.

The combination of thallium-silver with the calcium-lead produced anodes of apparently greater stability than the anodes of silver-calcium lead. The influence of thallium-silver, however, is seen to greatly neutralize the lowering effect of calcium on the anode potential. The figures show a 51 per cent. lowering in anode potential (below pure lead) for an anode composed of 0.1 per cent. Ca, 2 per cent. Tl, and only 21 per cent. lowering for an anode containing 0.75 per cent. Ag, but with the same calcium and thallium content.

The influence of tin is seen to almost wholly neutralize the lowering effect of calcium. This influence is seen in evidence in anodes 26 and 27, Table 1.

The electrolyte consisted of acid zinc sulfate solution containing approximately 68 g. Zn per liter and 200 g. H_2SO_4 per liter initially, and approximately 30 g. Zn per liter and 257 g. H_2SO_4 per liter at the end of each cycle. Manganese was kept approximately constant at 0.6 g. Mn per liter, in the form of sulfate.

TABLE 1.—*Anode Composition, Polarization and Lead Content of Cathode Zinc*

Current Density, 100 Amp. per Square Foot

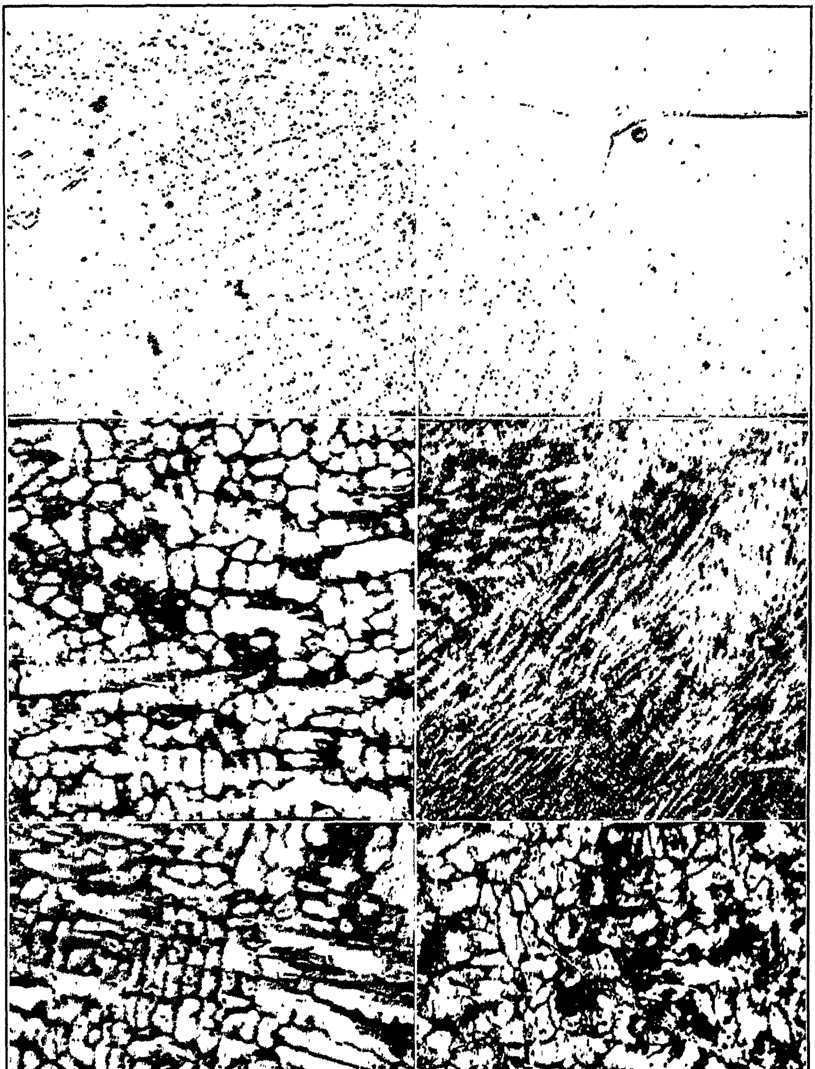
Anode No.	Intended Anode Composition, Per Cent.					Anode Polarization, Volts	Decrease in Potential below That of Pure Lead, Per Cent.	Lead in Cathode Zinc, Per Cent.
	Pb	Ca	Ag	Tl	Sn			
269D, 20	98.83	0.17	1.00			0.240	54.10	0.022
269E, 21	98.86	0.14	1.00			0.244	52.30	0.020
269F, 22	98.90	0.10	1.00			0.255	50.20	0.021
102, 23	97.90	0.10		2.00		0.250	51.15	0.040
103. 24	96.90	0.10		3.00		0.284	43.70	0.011
104. 25	95.90	0.10		4.00		0.310	39.45	0.008
120, 26	95.0				5.00	0.450	12.12	0.026
121, 27	94.90	0.10			5.00	0.443	13.47	1.000
105, 28	98.40	0.10	0.50	1.00		0.335	34.55	0.0175
106, 29	97.40	0.10	0.50	2.00		0.340	33.60	0.011
107, 30	98.15	0.10	0.75	1.00		0.349	31.82	0.014
108, 31	97.15	0.10	0.75	2.00		0.395	20.90	0.010
A	100–					0.512	00.00	0.064

1

2

3

FIG. 1.—ANODE 20. AG, 1 PER CENT., CA, 0.17 PER CENT.
FIG. 2.—ANODE 21. AG, 1 PER CENT., CA, 0.14 PER CENT.
FIG. 3.—ANODE 22. AG, 1 PER CENT., CA, 0.10 PER CENT.
All × 200.



4 5

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8 9

Fig. 4.—Anode 23. Tl, 2 per cent., Ca, 0.10 per cent.
Fig. 5.—Anode 24. Tl, 3 per cent., Ca, 0.10 per cent.
Fig. 6.—Anode 28. Tl, 1 per cent., Ag, 0.5 per cent., Ca, 0.1 per cent.
Fig. 7.—Anode 29. Tl, 2 per cent., Ag, 0.5 per cent., Ca, 0.1 per cent.
Fig. 8.—Anode 30. Tl, 1 per cent., Ag, 0.75 per cent., Ca, 0.1 per cent.
Fig. 9.—Anode 31. Tl, 2 per cent., Ag, 0.75 per cent., Ca, 0.1 per cent.

All × 200.

The lead contents of the cathodes shown in Table 1 are of interest in a relative sense, as all conditions were uniform. Other influences affecting anode corrosion or stability should affect these values in a similar manner.

Microscopic Study of Anodes

Figs. 1 to 9 show the structures of some of the anodes.[3] The thallium-calcium-lead alloys are of the solid solution type with a slight amount of compound precipitated. The silver-calcium-lead alloys and the silver-thallium-calcium-lead alloys are eutectiferous alloys. So far in this investigation we have been unable to find any relation between properties and structure.

[3] Photomicrographs by courtesy of the Metallographic Laboratory, Western Electric Co., Hawthorne Station.

Extraction of Tantalum and Columbium from Their Ores*

By Colin G. Fink† and Leslie G. Jenness,‡ New York, N. Y.

(New York Meeting, February, 1931)

Tantalum and columbium occur together in tantalite and columbite ores, which may be considered as ferrotantalate ($FeTa_2O_6$), with part of the iron and tantalum replaced by manganese and columbium respectively, the general formula, therefore, becoming $(Fe, Mn)O.(Ta, Cb)_2O_5$. The ratio of tantalum to columbium is not fixed; the amount of tantalum may vary from one-third to three times that of columbium. Minerals having a high content of tantalum are called tantalite, while those possessing larger amounts of columbium are classified as columbite ores. These minerals are found in pegmatites and often occur in veins associated with cassiterite and wolframite, and may contain small amounts of tin, tungsten, titanium and silica.[1]

Methods of Extraction

Several methods have been proposed for the extraction of tantalum and columbium from their ores, among them being:

1. *Fusion with Potassium or Sodium Bisulfate.*—The finely ground ore is fused with potassium bisulfate, after which the mass is boiled with water and the fusion repeated several times. The residue finally resulting is digested with ammonium sulfide to remove traces of tin and tungsten, boiled with hydrochloric acid, filtered and washed. These oxides are then dissolved in hydrofluoric acid, the silica evolved as silicon tetrafluoride and the hydrofluoric acid solution used for the potassium double fluoride separation of tantalum.

2. *Fusion with Sodium Carbonate and Sodium Nitrate.*[2]—This fusion renders the tantalum and columbium soluble, and they are extracted

* From the dissertation by L. G. Jenness submitted in partial fulfilment of the requirements for the degree of Doctor of Philosophy in the Faculty of Pure Science of Columbia University, New York City.

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[1] U. S. Bur. Mines, *Min. Resources of the U. S.* 1927 (1929) Pt. I, 406–413. See also G. W. Sears: Progress in Production and Use of Tantalum. *Trans.* A. I. M. E., Genl. Vol. (1930) 317 and M. M. Austin: Working Properties of Tantalum. *Trans.* A. I. M. E., Inst. Met. Div. (1930) 551.

[2] E. Wedekind and W. Maass. Über die Darstellung von Tantalsäure aus westaustralischen Fergusonit und über Natriumtantalat. *Ztsch. f. angew. Chem.* (1910) **23**, 2314.

with a small quantity of soluble impurities. The tantalum, columbium and titanium are the only metals present which are precipitated from this solution by sulfur dioxide, thus enabling the separation of these metals from the remainder of the ore.

3. *Volatilization with Chlorine.*—H. S. Cooper[3] states that tantalum, columbium and iron can be volatilized as chlorides by passing chlorine over the ore plus carbon at 500° C. The chlorides are then hydrolyzed in a sodium chloride solution, to precipitate tantalic and columbic acids and retain the iron as soluble ferric chloride.

4. *Fusion with Sodium Hydroxide.*—The ore is fused with sodium hydroxide, the fused mass leached with sulfuric acid to dissolve manganese and iron, and the remainder of the treatment continued as described under method 1. This method is preferred to that of the bisulfate fusion, since it can be carried out in iron.

A review of these methods, as well as the prevailing price of tantalum oxide, encouraged us in our search for a simpler and cheaper method. This led to a study of the leaching of tantalum and columbium ores with aqueous solutions comprising reagents readily obtainable at low cost.

Preliminary Leaching Results

It soon became evident that a leaching solution for Ta, Cb ores must be capable of decomposing the ore, as well as acting as a solvent for the two metals, tantalum and columbium. Hydrofluoric acid and carboxylic acids are the only commercial, cheap acids that dissolve tantalum oxide. Upon trial we found that these two acids will decompose the ore, hydrofluoric acid being, however, much more effective. An important turn in our research followed the discovery that, when hydrofluoric acid and oxalic acids were used together, the ore was attacked much more readily than when either acid was used alone. Accordingly, investigation was then directed toward a study of:

1. The effect of carboxylic acids.
2. The optimum conditions for leaching, using both acids.
3. A method of recovering tantalum and columbium from the solution after leaching.
4. The efficiency and cost of extraction by our new method.

Materials Used

The materials used for this work were of two classes. Baker's analyzed chemicals were used in making a study of the effect of carboxylic acids on the hydrofluoric acid leaching solution, while com-

[3] H. S. Cooper: U. S. Patent 1507987 (Sept. 9, 1924).

mercial chemicals were employed in studying the process on a larger scale of production. The calcium fluoride, used as a source of hydrofluoric acid, was the native ore, fluorspar. A 200-mesh tantalite ore was procured from the Foote Mineral Co., Philadelphia, with the following analysis: Ta_2O_5, 55.0 per cent.; Cb_2O_5, 22.4; SiO_2, 10.0; TiO_2, 0.39; SnO_2, 2.29; Fe_2O_3, 0.18; MnO_2, 9.63.

Leaching by the Steam-dilution Method

Combined Effect of Hydrofluoric and Oxalic Acids

A series of leachings was made on 100 g. of tantalite ore, the charges consisting in turn as follows:

(*a*) 100 g. 50 per cent. HF.
(*b*) 310 g. $H_2C_2O_4.2H_2O$.
(*c*) 100 g. 50 per cent. HF and 310 g. $H_2C_2O_4.2H_2O$.
(*d*) 100 g. CaF_2 and 200 c.c. conc. H_2SO_4.
(*e*) 100 g. CaF_2 and 310 g. $H_2C_2O_4.2H_2O$.
(*f*) 100 g. CaF_2, 310 g. $H_2C_2O_4.2H_2O$ and 200 c.c. H_2SO_4.

These leachings were conducted in a calibrated, lead-lined pail of 8 liters capacity. Steam was admitted through a lead coil with small perforations in the lower turn. The condensation of the steam maintained a temperature of about 95° C. Sufficient water was added to the original charge to give a starting volume of 500 c.c. The volume of the solution was increased, by the condensation of the steam, as the leaching continued, about 4 hr. being required to bring the volume to 6 liters.

Small samples were withdrawn periodically as the volume of the leaching solution increased, and were analyzed for total oxides by precipitation with ammonium hydroxide. The results are expressed graphically in Fig. 1 in terms of ore leached. Chemical analysis showed that the ratio of tantalum to columbium in the leaching solution was nearly the same as in the ore. In comparing the results (Fig. 1), it is evident that the combined effect of hydrofluoric acid and oxalic acid is far greater than that of either reagent used singly. This is probably due to the fact that hydrofluoric acid is a much better decomposing agent for the ore, while the oxalic acid acts as a much better solvent. Furthermore, actual extractions (uppermost curve "observed") are almost twice the "calculated."

It is evident also that the rate of decomposition of the ore is largely dependent upon the concentration of hydrofluoric acid. This is shown by a comparison of the leaching with hydrofluoric acid and that using calcium fluoride and sulfuric acid. The concentration of hydrofluoric acid is much greater at the start in the former case, while its rate of volatilization, and, consequently, loss from solution, is greater. It would

be expected, therefore, that the rate of leaching would be much greater over the initial period by the introduction of hydrofluoric acid, and that it would be greater over the final period when using calcium fluoride and sulfuric acid. It is also probable that some of the difference between these two curves is explained by the increased solubility of the tantalum and columbium compounds in the presence of strong acids.

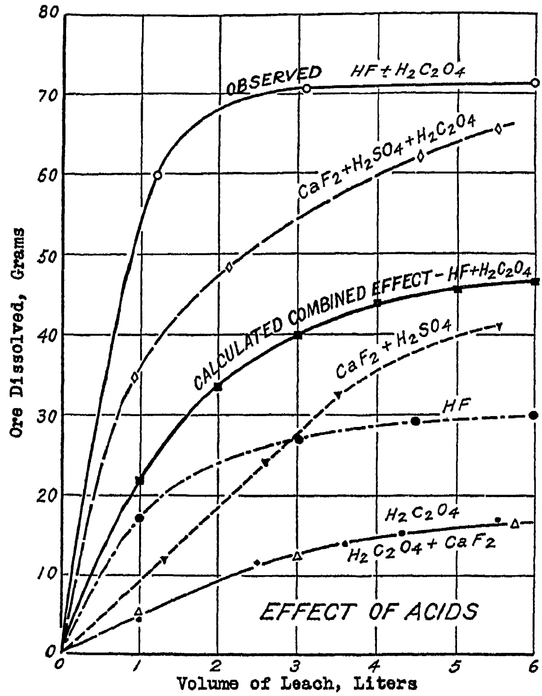

FIG. 1.—EFFECT OF ACIDS.

Effect of Concentration of Oxalic Acid

To determine the effect of varying concentrations of oxalic acid, leachings were conducted with one-half and one-quarter the amount originally selected, 310 g., the amount of calcium fluoride and sulfuric acid remaining constant. The results of these experiments are shown in Fig. 2.

From the curves of Fig. 2, it is noticed that when the amount of oxalic acid is reduced from 310 to 155 g. the quantity of ore leached remains the same until the volume reaches 2.5 l., after which it gradually falls off. When the quantity of oxalic acid is again halved, the amount of ore leached diminishes markedly. It seems evident that an excess of oxalic acid, over that necessary to form the oxalate compounds, is necessary to hold the Ta, Cb compounds in solution. This is also verified by

Powell and Schoeller[4] who have worked with a solution of ammonium oxalate and sulfuric acid as a solvent for tantalum and columbium oxides.

Since little improvement in leaching is shown by a solution containing more than 155 g. of oxalic acid, and since smaller quantities show a decided drop in leaching efficiency, it is concluded that a final concentration of 30 g. per liter would constitute the most economically efficient leaching solution.

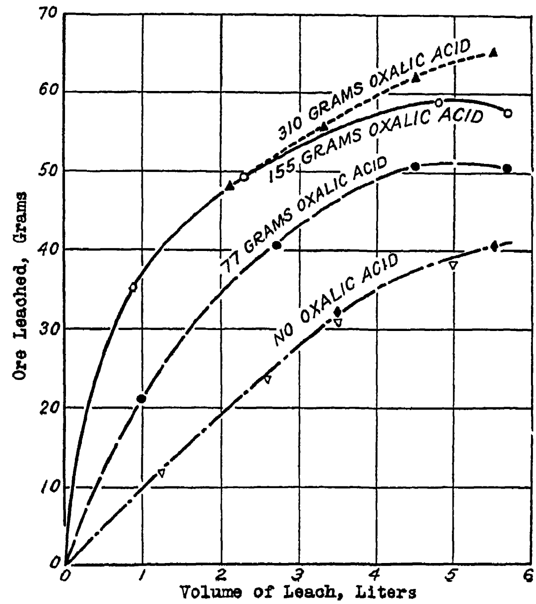

FIG. 2.—EFFECT OF OXALIC ACID WHEN ADDED TO HYDROFLUORIC.

EFFECT OF OTHER CARBOXYLIC ACIDS

After recognizing the beneficial effect of oxalic acid upon a hydrofluoric acid leaching solution for these ores, it seemed of value to determine the definite cause for this effect. It was found that the presence of reducing gases, such as carbon monoxide and sulfur dioxide, did not exhibit this effect, and it was concluded that it could not be attributed to any decomposition of the oxalic acid. Furthermore, the total amount of oxalate added was shown by analysis to be present after leaching was complete. It seemed, therefore, that its function must be that of a solvent, and a study of other carboxylic acids was made.

Accordingly, a series of leachings was conducted using molecularly equivalent quantities of other carboxylic acids. The leach obtained with

[4] A. R. Powell and W. R. Schoeller: Investigation into the Analytical Chemistry of Tantalum, Niobium and Their Mineral Associates, IV. A New Method for the Separation of Tantalum from Niobium. *The Analyst* (1925) **50**, 485.

100 g. of calcium fluoride, 200 c.c. sulfuric acid and 155 g. of oxalic acid was taken for comparison. The curves plotted in Fig. 3 for molecularly equivalent quantities of the acids—succinic, $CH_2COOH.COOH.CH_2$; tartaric, $(CH)_2(OH)_2(COOH)_2.H_2O$; citric, $(CH_2)_2(COOH)_2.CH.OH.$-$COOH$—selected to replace the oxalic acid in successive leachings.

We observed that every one of the carboxylic acids tried, when added to the hydrofluoric acid solution, increased the rate of solution of the ore during the initial part of the leaching,—that is, when the solutions were concentrated. It was of further interest to note that the acceleration effect decreased with an increase in the molecular weight of the acid,

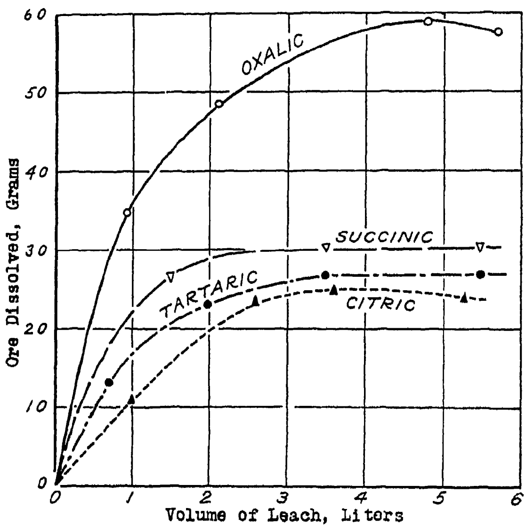

FIG. 3.—EFFECT OF CARBOXYLIC ACIDS.

and that the presence of the hydroxyl group (in tartaric and citric acids) appeared to have no decided influence on the rate of solution. On the basis of our tests we believe it is the carboxylic group, COOH, which is the important constituent of the various acids tested, that renders the tantalum and columbium more soluble, and that oxalic acid is the preferred acid because in it the carboxylic group is most active.

Action on Columbite vs. Action on Tantalite

To determine the relative decomposing action of hydrofluoric acid on columbite and on tantalite ores, a leaching was conducted on a sample of columbite obtained from the Foote Mineral Co. This leaching was made with 100 g. of calcium fluoride and 200 c.c. of sulfuric acid. Fig. 4 shows the results, compared with those from the tantalite ore.

It is evident that there is little difference in the rate of decomposition of columbite and tantalite by the use of hydrofluoric acid. Since columbium oxide is even more soluble in oxalic acid than tantalum oxide, and since the ore must be decomposed before the columbium can be dissolved, it seems certain that all of the leaching results obtained with tantalite will be equally applicable to columbite.

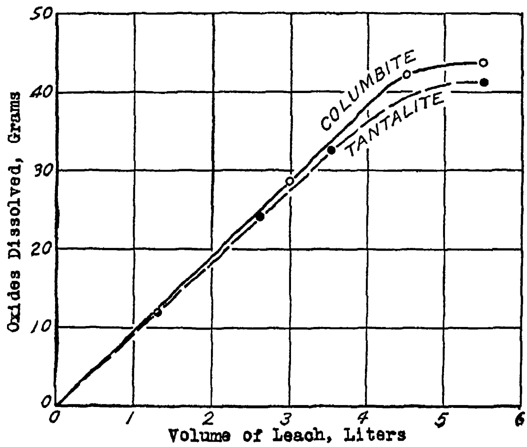

FIG. 4.—COMPARISON OF ACTION ON TANTALITE AND COLUMBITE.

LEACHING AT CONSTANT VOLUME

Effect of Acid Concentration

In these experiments the method of leaching was to allow the condensation of steam within the leaching solution, this steam serving as a source of heat, but at the same time decreasing the concentration of the leaching reagents through dilution of the solution. The leaching curves previously considered have, as a result, three variables—time, acid concentration and amount of ore leached. Since the solution is continuously increasing in volume with time, the concentration of ore within the solution would not be expected to be in equilibrium with a given acid concentration.

A consideration of this fact would lead to the belief that if the volume of the solution is maintained constant, other conditions remaining the same, the amount of ore leachable by a given acid concentration should be increased. With this in view, a series of constant-volume leachings was conducted with several different acid concentrations, but always with the ratio of sulfuric acid to calcium fluoride to oxalic acid the same. This ratio was taken as 200 c.c. sulfuric acid, 100 g. calcium fluoride and 155 g. of oxalic acid, the same ratio as that used in the steam condensation leaching.

Thus the H_2SO_4 concentration was equivalent to 80, 160, 250, 320 and 400 c.c. per liter in the first five experiments in this series. In a sixth experiment the concentration of H_2SO_4 was again 400 c.c. per liter but the oxalic acid was omitted.

Rate of Leaching: Constant Solution Volume

Fig. 5 shows the rate of leaching at constant volume, and the maximum concentration of dissolved oxides obtained, with and without the presence of oxalic acid. It is obvious that oxalic acid shows the same beneficial effect when leaching at constant volume as when leaching by the

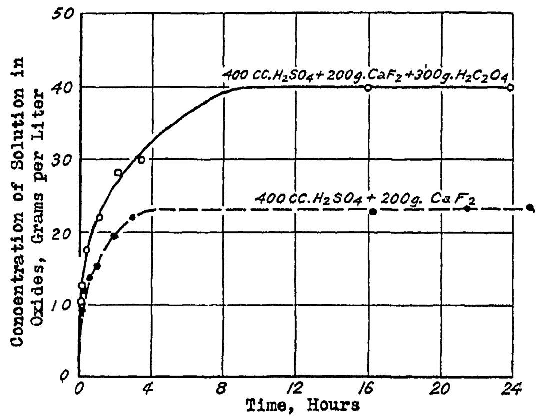

FIG. 5.—RATE OF LEACHING AT CONSTANT VOLUME.

steam-dilution method. In plotting our other results with constant solution volume, we obtain the same type of curve as in Fig. 6, differing only in the maximum concentration. How this maximum varies directly with the concentration of acids taken is shown in Fig. 7.

It is of interest to note that the rate of approaching the maximum concentration appears to be the same for all concentrations when oxalic acid is used, but different without the presence of oxalic acid. The logarithmic curves in Fig. 6 show that this rate can be represented by the equation

$$t = a(c/C)^b$$

where t = time in hours; c = concentration in grams per liter of dissolved Ta, Cb oxides at any time t; C = maximum concentration in dissolved oxides expressed as grams per liter; a = constant and b = constant.

It is realized that there are a large number of reactions taking place during the leaching, but it is believed that the reactions governing the rate of leaching can be represented as taking place in the following manner:

$$FeTa_2O_6 + 12HF = 2TaF_5 + FeF_2 + 6H_2O \quad [1]$$
$$2TaF_5 + 5H_2C_2O_4 = Ta_2(C_2O_4)_5 + 10\ HF \quad [2]$$

The formation of a true oxalate may be somewhat questionable however, but a reaction similar to this must take place. It is also under-

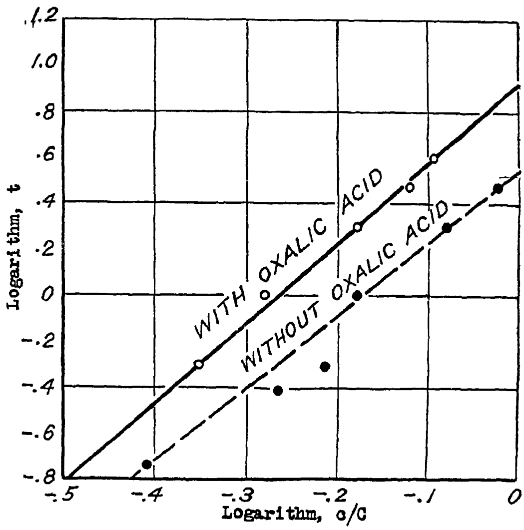

FIG. 6.—LOGARITHMIC CURVES. RATE OF APPROACHING LIMITING CONCENTRATION.

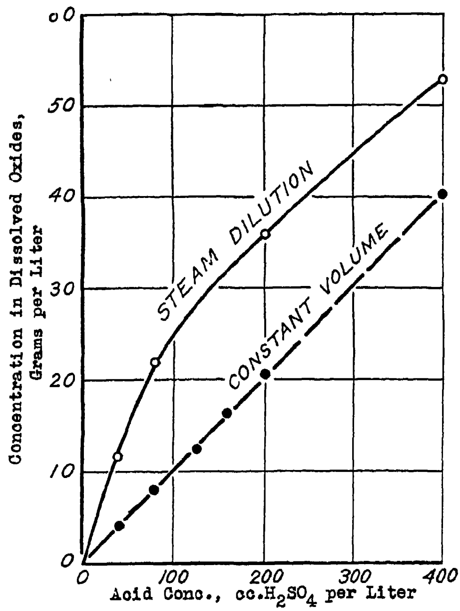

FIG. 7.—DISSOLVED OXIDES AS FUNCTION OF ACID CONCENTRATION.

stood that columbium and manganese may replace the tantalum and iron respectively in the above reactions. Our experimental data indi-

cated that the decomposing reaction takes place at a faster rate than does the transfer to the oxalate.

Comparison of Results by the Two Leaching Methods

By comparing the concentration of dissolved oxides for a given acid concentration by the two methods of leaching (Fig. 7), it will be observed that the steam-dilution method is far superior to the constant-volume method. This is contrary to what might be expected, but can be explained by the observation that the hydrofluoric acid-oxalic acid solution is capable of retaining more ore in solution than can be decomposed for any given acid concentration. This fact has been repeatedly observed throughout our investigation, and seems to be due to the manner in which the oxalic acid functions.

We found that the rate of decomposition of the ore is greater than the transfer to the soluble oxalate compound. We also found experimentally that the two reactions could not be carried out singly in succession, with results the same as those obtained when the two reactions are allowed to take place simultaneously. The explanation, therefore, would seem to be as follows:

1. When the two reactions are taking place simultaneously the hydrofluoric acid concentration is kept higher, due to its liberation in the second reaction. Therefore, the amount of ore decomposed will be increased.
2. The rate of the second reaction is slower, and the maximum benefit of the oxalic acid cannot be obtained, since there is another factor to be considered; namely, the rate of volatilization of the hydrofluoric acid.
3. Best results are obtained if the second reaction takes place with speed at least equal to that of the decomposing reaction.
4. The nearest approach to this ideal condition is to allow the reaction to take place in concentrated solution, where the extent of the reaction is greatly magnified, and to dissolve the soluble oxalate compound by dilution of the solution in the presence of free oxalic acid. This is what takes place in the steam-dilution method.

Optimum Conditions for Leaching

The experiments showed that 155 g. of oxalic acid in 5 l. was practically as efficient as larger concentrations of acid, but that smaller concentrations caused a drop in leaching efficiency. This concentration of acid will leach about three times as much ore by the steam-dilution method. The best conditions for leaching, according to our method, therefore, are as follows: Steam dilution; an excess of fluorspar with the ore; 200 c.c. of concentrated sulfuric acid, and 150 g. of oxalic acid present for each 5 l. of final leaching solution.

Recovery of Tantalum and Columbium from Leaching Solution

Crystallization of Potassium Tantalofluoride

The only utilized commercial method of separating tantalum from columbium is based upon the difference in solubility of tantalum and columbium double fluorides of potassium, the former being much less soluble. It seemed of value, therefore, to investigate the possibility of separating the tantalum from the leaching solution by means of this principle.

For this study a leach was conducted in a steam-jacketed lead evaporator of about 19 l. capacity. The final solution of 19 l. contained 190 g. of dissolved Ta, Cb oxides. The clear solution was transferred to a second evaporator of the same type, heated to about 80° C., and 150 g. of potassium fluoride and 300 g. of 50 per cent. hydrofluoric acid was added. Considerable crystallization took place immediately, and this increased upon concentration of the solution. Some of the first crystals were removed, after which the solution was evaporated to 5 l., the crystals filtered off, washed and dried over a steam radiator.

The weight of the final product was 960 g. and analyzed 7.59 per cent. tantalum oxide. The presence of iron could be faintly detected with potassium sulfocyanide and manganese by the sodium carbonate-potassium chlorate bead, but both elements were present only in faint traces. The presence of columbium could not be detected by reduction with tin or by the method of Powell and Schoeller.[5] It seemed evident, therefore, that these crystals consisted of oxalic acid and potassium tantalofluoride.

On the assumption that the dissolved oxides contained 60 per cent. tantalum oxide, the tantalum recovered free from columbium was about 64 per cent. of that leached. It would be necessary, however, to separate the large amount of oxalic acid accompanying it. This could be done by further crystallization from hot solutions, and a recovery of considerable oxalic acid could be made.

The first crystals removed were of high purity and contained no oxalic acid. No data are available to show the percentage of recovery of the pure product, but it is certain that considerable can be recovered which will not require recrystallization.

Fractional Precipitation with Alkalies

It was observed that tantalum and columbium could be precipitated from the leaching solution while it was still acid, and this led to an investigation of this method of recovering the oxides of these metals. An electrometric titration of the solution with ammonium hydroxide, using a quinhydrone electrode, showed precipitation to start at a pH

[5] A. R. Powell and W. R. Schoeller: *Op. cit.*

of 1.1. Further study showed that a large amount of precipitate was obtained when adding ammonium hydroxide to the solution until basic to methyl orange, pH of 5, and considerably more when basic to methyl red, pH of 6. Neither of these precipitates showed the presence of iron or manganese when testing with potassium sulfocyanide, and the sodium carbonate-potassium chlorate bead respectively. Experiments were then conducted to determine the efficiency of recovery and the purity of the product.

Using sodium hydroxide as the precipitant, and precipitating just to the acid side of litmus paper, a product was obtained, after drying at 100° C., which was soluble in hydrochloric acid. The product contained no metals of the acid hydrogen sulfide group, no titanium, showed a faint test for iron and manganese, and contained 53.8 per cent. of combined tantalum and columbium oxides. It was evident, however, that it contained a large amount of oxalate, sodium and water, and appeared to be a double oxalate of the metals with sodium.

Since it was evident that the alkali would accompany the metal in the final product, it seemed advisable to use ammonium hydroxide as the precipitant, which later could be eliminated by ignition. A precipitation was conducted, therefore, by adding ammonium hydroxide until the solution was just basic to litmus paper, and then sulfuric acid until it was distinctly acid to litmus paper. The precipitate was filtered off, after washing several times by decantation, and dried at 110° C. The decanted solution and washings were made basic with ammonium hydroxide, filtered, washed and dried. Each of these products was then weighed and analyzed. The results are shown in Table 1.

TABLE 1.—*Weights and Analyses of Products*

	Grams	Per Cent.
No. 1 Precipitate on acid side of litmus	32.0	86.5
No. 2 Precipitate on basic side of litmus	5.0	13.5

Analyses	No. 1 Per Cent.	No. 2 Per Cent.
Ta_2O_5	44.8	6.2
Cb_2O_5	15.2	
SiO_2	0.81	39.9
TiO_2	none	
H_2S (acid) metals	none	
Fe_2O_3 (KCNS test)	trace	
MnO_2 (Na_2CO_3—$KClO_3$ bead test)	trace	
C_2O_4	16.5	
NH_3	present	
SO_3	0.33	
F	present	

Recovery of $Ta_2O_5 + Cb_2O_5$

	Grams	Per Cent.
$Ta_2O_5 + Cb_2O_5$ in No. 1	19.2	98.5
$Ta_2O_5 + Cb_2O_5$ in No. 2	0.31	1.5

It was evident that the tantalum and columbium could be precipitated from the acid solution in a state practically free from other metals, and with very good recovery of the metal oxides.

Selected Method for Commercial Operation

The method utilized commercially would depend upon the product desired. For purposes where no separation of tantalum and columbium is desired, the fractional precipitation with ammonium hydroxide would be employed. The successive use of these two methods, however, could be used to recover a large amount of the tantalum free from columbium and the remainder with columbium.

Semitechnical Results

For a study of the method of leaching on a larger scale it was considered advisable to utilize ores that were available in this country. The ores studied were samarskite and columbite obtained from A. D. Mackay, New York City, the former stated to be from North Carolina and the latter from South Dakota.

Samarskite

Samarskite is very troublesome to work for tantalum and columbium by present methods, as it contains uranium, cerium, yttrium, erbium and other rare earths in addition to the metals present in tantalite and columbite. The tantalum and columbium content is only 50 to 55 per cent. of the ore, represented as oxides, and the yield, therefore, is much smaller for this ore. It was realized, however, that the rare earth metals should not be leached in the presence of oxalates and fluorides and could, consequently, be left in the gangue. Except for uranium, therefore, there would be no additional difficulty in working with this ore.

To verify this assumption, a 19-l. leach was made on this ore, using the following materials: 500 g. ore, 600 g. fluorspar, 1200 g. oxalic acid, 1200 c.c. sulfuric acid. A volume of 18 l. obtained from the leach showed a concentration of 18.2 g. per liter of oxides precipitable with ammonia, showing a leach of 338 g. of oxides. It is evident from these results, as well as from other observations, that this ore is more easily leached than tantalites and columbites.

The tantalum and columbium were precipitated from the solution with ammonium hydroxide, keeping the solution acid to litmus, washed by decantation and finally filtered. A sample dried at 110° C. showed the following analysis:

	Per Cent.		Per Cent.
Loss on ignition	36.5	UO_3	present
$Ta_2O_5 + Cb_2O_5$	55.0	Rare earths	none
Fe_2O_3	trace	C_2O_4 (oxalate)	24 1
MnO_2	trace	SO_3	0.20
NH_3	present	SiO_2	1.6

Although no further work was done with this ore, it is believed that the uranium could be removed by leaching the precipitated oxides with ammonium carbonate.

Columbite

The columbite ore was reduced from the lump to a 50-mesh material and then leached. The leach was conducted in a 189-l., lead-lined tank by the steam-dilution method, the charge consisting of 6 lb. (2.7 kg.) of ore; 10 lb. (4.5 kg.) fluorspar; 12 lb. (5.4 kg.) oxalic acid; 32 lb. (14.5 kg.) 66° Bé. sulfuric acid; 5 gal. (19 l.) of water.

Steam was introduced through a rubber hose, and the leach conducted over a period of 5 hr., a final volume of 189 l. being obtained. The solution was allowed to settle overnight, and the clear solution was then transferred to a wooden tank for precipitation.

The wooden tank had a diameter of 2 ft. (0.61 m.), 1 in. (2.5 cm.) of depth, containing 1.955 gal. (7.4 l.). The tank was filled to a depth of 23 in. (58.4 cm.), representing 45 gal. (170 l.) of solution. This volume was nine-tenths of the leach, the balance being saved for other investigations. The oxides were precipitated with 26° Bé. ammonium hydroxide to the acid side of litmus paper, 40 lb. being required. The precipitate was washed twice by decantation and filtered.

Since it was desired to use these oxides for further investigations, and since drying rendered them less soluble, a sample was analyzed for purity, and the remainder was stored in well-stoppered bottles. The yield was then obtained by determining the total oxides leached and the oxides not precipitated by ammonia on the acid side of litmus. The results are shown in Table 2. The sample dried at 110° C. showed 1.3

Table 2.—*Oxide Precipitation*

Concentration of leaching solution in total oxides, g. per l.	10.9
Total oxides in 189 l. of solution, kg	2.06
Volume of solution after precipitation, l	191.5
Concentration of solution after precipitation in total oxides, g. per l	1.83
Total oxides not precipitated from 153 l., kg	0.35
Total oxides not precipitated from 189 l., kg	0.39
Total oxides precipitated from 189 l., kg	1.67

per cent. silica and faint traces of iron and manganese. It would be concluded, therefore, that about 3.5 lb. (1.6 kg.) of combined tantalum and columbium oxides could be recovered after calcining the product.

Cost Calculations

On the basis of the semitechnical results, the cost of the materials consumed for the production of 1 lb. (0.45 kg.) of the combined oxides of tantalum and columbium would be:

10 lb. fluorspar at 1 c. per lb.	$0.10
12 lb. oxalic acid at 11 c. per lb.	1.32
32 lb. 66° Bé. sulfuric acid at 0.8 c. per lb.	0.26
40 lb. 26° Bé. ammonia at 3 c. per lb.	1.20
Total cost for 3.5 lb. oxides	$2.88
Cost per pound of oxides	0.82

The apparatus employed would be of a simple type, consisting of lead-lined wooden tanks. The cost of production, therefore, would depend largely upon the value of the ore worked, whether a separation of tantalum from columbium was required, and whether oxalic acid was recovered from the solution.

Since the cost of materials for leaching is only 48 c. per pound of combined oxides, and since it has been shown that at least 60 per cent. of the tantalum can be recovered directly from the leach in the form of potassium tantalofluoride, this would place the cost of leaching the recovered tantalum oxide at $1.14 per pound. Assuming only a 50 per cent. recovery of the oxalic acid during crystallization, the cost would be reduced to 63 c. per pound for the materials. It seems fair to estimate, therefore, that a pure grade of tantalum oxide could be produced commercially at a cost of $2 per pound, and that this would make no allowance for an equivalent quantity of the combined oxides which could be recovered from the solution at a considerably cheaper cost. In view of the fact that it is difficult to procure an appreciable quantity of tantalum oxide on the market, even at a price of $30 per pound, it would seem that this method warranted consideration.

Summary and Conclusions

A method of leaching tantalum and columbium ores has been developed and the optimum conditions determined. Study has been devoted to the recovery of tantalum in the form of potassium tantalofluoride and to the recovery of the combined oxides of tantalum and columbium from the solution after leaching. Tantalite, columbite and samarskite ores were investigated. On the basis of our experimental results we conclude that:

1. Hydrofluoric acid acts as a good decomposing agent for the tantalum and columbium ores.

2. The presence of carboxylic acids greatly increases the rate of decomposition and solution of the ore.

3. The effectiveness of the carboxylic group decreases with increased molecular weight of the acid containing the group, oxalic acid being by far the most efficient.

4. Hydrofluoric acid can be produced during the leaching, by employing fluorspar and sulfuric acid.

5. The combination of hydrofluoric and oxalic acids is capable of retaining more tantalum and columbium in solution than it can decompose from the ore at a given acid concentration.

6. This makes it more feasible to start with a high concentration of acids and utilize steam condensation for heating and diluting the solution.

7. The quantity of materials selected for leaching should be such that an excess of fluorspar is present, and 30 g. of oxalic acid and 40 c.c. of sulfuric acid per liter of final solution.

8. About 60 per cent. of the tantalum can be recovered in the form of potassium tantalofluoride directly from the solution after leaching, but the crystals will contain a large amount of oxalic acid.

9. By recrystallization of the above product, a large amount of oxalic acid can be recovered for leaching, and a pure salt of tantalum can be prepared.

10. The combined oxides of tantalum and columbium can be recovered from the solution by fractional precipitation with ammonium hydroxide.

11. The cost of producing Ta, Cb oxides by our method makes the process look very attractive in view of present prices.

Application of the Wire Saw in Marble Quarrying

By W. M. Weigel,* St. Louis, Mo.

(New York Meeting, February, 1930)

The first successful use of the wire saw in slate quarrying in the United States was late in 1926, at the quarry of the Colonial Slate Co. near Wind Gap, Pa. This installation was sponsored and supervised by the United States Bureau of Mines in collaboration with the National Slate Association.[1] The success of this trial resulted in other installations of wire saws, so that 27 were in operation at Pennsylvania slate quarries[2] by October, 1928.

Marble Quarrying with a Wire Saw

Until recently, little or no attempt has been made to use the wire saw in the quarrying of other stone, but it is believed that its successful use in slate will be followed by attempts to take advantage of its possibilities in other quarrying operations. An example of such a trial is the recent installation of a wire saw at the marble quarry of the Saint Clair Marble Co. near Guion, in the Ozark region of northern Arkansas. This quarry is in a comparatively new district and the decision to open and develop it had to be made in the face of many unknown factors. For this reason, it was essential that the development costs be held at a minimum, and as a wire saw could be installed more cheaply than standard channeling equipment, it was decided to try it out. One factor largely controlling the decision was that the location was peculiarly adapted to this method. The point of development selected was on the rather abrupt end of a ridge formed on the south by an almost perpendicular bluff along the White River and on the north by a rather steep slope with little or no overburden. By taking advantage of this topography it was possible to place the supporting standards for the sheaves at both the outer and driving end without the necessity of drilling or excavating openings to receive them, and to locate the motive power in line with these standards, which made it unnecessary to deflect the sheaves.

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* Mineral Technologist, Missouri Pacific Railroad Co.

[1] O. Bowles: The Wire Saw in Slate Quarrying. U. S. Bur. Mines *Report of Investigations* 2820 (1927).

[2] O. Bowles: The Wire Saw in Slate Quarrying. U. S. Bur. Mines *Report of Investigations* 2918 (1929).

The point of development is adjacent to the right of way of the Missouri Pacific R. R., so that by using a short spur track parallel to the face of the cliff it is possible to load the quarry blocks directly on to railroad cars by the main quarry derrick. The foot of this derrick is on a rock bench 30 ft. above the level of the railroad.

Rock Formations

The rocks exposed from the level of White River upward are of Ordovician age.[3] About 30 ft. of Plattin limestone is exposed at the bottom, and the top of this formation constitutes the present quarry floor. Above this there is about 12 ft. of the Kimmswick formation and above this 90 to 100 ft. of the Fernvale. Both the Fernvale and Kimmswick furnish the marble quarried. These two formations are similar in texture, being medium hard and rather coarsely crystalline. The prevailing color is light gray; certain parts show a pink tint, and others fine veinings of various degrees of blue to darker gray. The bedding is approximately horizontal.

Wire Saw Equipment

The sheave support and auxiliary equipment were made and supplied by a machinery manufacturer at Joplin, Mo. The motive power consists of a four-cylinder automobile engine removed from a Dodge car and placed on suitable skids. In the design of the equipment simplicity was the first consideration, as the installation was more or less of an experiment; automatic feeds and such other improvements were dispensed with. The guiding sheaves supporting the wire at the exit end of the cut are mounted on an 8-in. H-column. The top guiding sheave is stationary and the lower one is mounted on a movable slide, which can be lowered to feed the wire into the cut by an arrangement of a cam and clamp. The foot of this column is a steel plate 18 in. square, which is anchored to the rock by expansion anchor bolts. As the outer end of the block being sawed is the cliff face, the supporting sheaves at this end are mounted on an angle-iron frame hanging down over the face of the cliff and supported by an angle-iron bracket carried back about 8 ft. on to the quarry floor and anchored with expansion bolts on each side of the cut. The top sheave is fixed at the top of this frame and the lower one is carried on a small four-wheel carriage, which can be moved up or down the angle-iron frame. This carriage is supported at the top and bottom by a flexible wire cable, which passes over small sheaves at the lower and upper ends of the frame and thence to a small winch. The wire is lowered

[3] H. D. Miser: Deposits of Manganese Ore in the Batesville District, Arkansas. U. S. Geol. Survey *Bull.* 734 (1922).

against the bottom of the cut by means of this winch and the supporting cables of the sheave carriage. The general arrangement of the sheaves and supporting framework is shown in Fig. 1, and the general layout and

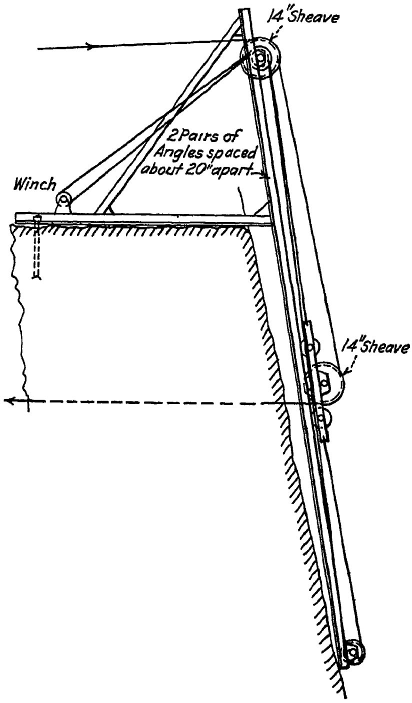

Fig. 1.—General arrangement of sheave support on cliff face and method of advancing wire into cut.

arrangement of the wire saw is shown in Fig. 2. Tension on the wire is maintained by a weighted car on the slope of the hill across a narrow valley. By this arrangement, a length of wire sufficient to complete a cut

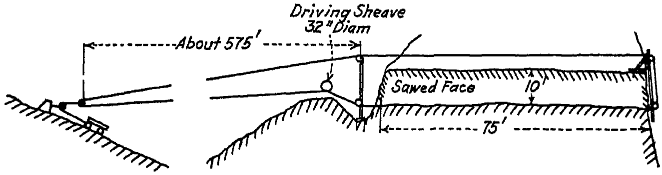

Fig. 2.—General arrangement of wire saw.

can be put into service. The driving sheave is 32 in. dia.; it is mounted on the same framework as the gasoline engine and is driven from it by a belt. The wire makes one complete turn around the drive sheave. The guide

sheaves are 14 in. dia. This is considerably smaller than at slate quarries, but so far the excessive bending has not caused the wire to break.

The sawing wire is ¼ in. dia., made up of three wires twisted together. The first one used was the same grade as that employed in sawing slate. The next one purchased was made from a little harder steel and apparently gave increased wear.

Operation of the Wire Saw

To begin a cut, the standard supporting the sheaves at the exit end is anchored in position, then the frame is placed to support the sheaves on the cliff face. The platform on which are mounted the gasoline engine and driving sheave is then lined up in position, and lastly the distant

Fig. 3.—Looking along finished cut after removal of quarry blocks.

tension sheave is suitably anchored. Made easy by the prevailing topography, these operations are all inexpensive and accomplished without difficulty. So far, a wire speed of about 20 ft. per sec. has been used, which is considerably faster than the practice in the Pennsylvania slate quarries. Other speeds have not been tried, so that it may or may not be the most efficient. The sand is fed by hand. A rather clean river sand has been used, but probably is too fine for the best results, as it would all pass a 20-mesh sieve. The sand grains were extremely round. A trial is to be made with fine angular flint grains.

After the saw is placed in operation, one man is sufficient to look after it. His duties consist of feeding the sand, maintaining the proper downward travel of the wire and the necessary oiling.

The tension sheave is placed at such a distance that about 1400 ft. of wire is employed. This length of wire lasts sufficiently long to complete any cut that has yet been attempted.

A tendency was noted in a long cut to leave the bottom at the center of the cut considerably higher than the ends. In one case the center was from 12 to 15 in. higher.

Fig. 3 shows a finished cut with the quarry blocks partly removed.

Costs and Rate of Cutting

The costs shown in Table 1 are those of one particular cut, which was 75 ft. long and 10 ft. deep, made in the Kimmswick formation. These are direct costs only and do not include the so-called overhead expenses of supervision, taxes, etc., or any depreciation on the equipment.

Table 1.—*Costs of One Particular Cut in Marble Quarrying*

Area of cut, 10 by 75 ft. = 750 sq. ft.

Time required, 70 hr., or 10.7 sq. ft. per hour.

1400 ft. ¼-in. wire strand	$26.15
1 man, 70 hr. at 25c	17.50
3 men, setting up 1 day, 30 hr. at 25c	7.50
4 tons sand at $1	4.00
70 gal. gasoline at 20c	14.00
Water supply, estimated	2.50
Total direct cost of 750 sq. ft.	$71.65
Cost per sq. ft.	0.0955

Conclusions

This installation is extremely simple and requires a minimum amount of supervision. It is still in an experimental stage, as up to date (October, 1929) only four cuts have been completed, so that while it seems to be successful it is not safe to say that it is entirely satisfactory. Conditions are ideal in this particular place, as the quarry face is open at both ends, so that there is no expense of preparing openings for the sheave standards. As the quarry face advances, the cuts will become much longer, and it is questionable whether a wire of the necessary length can be maintained under sufficient tension to complete the cut at the center to something near the same depth as at the ends, within a reasonable time, and also have sufficient reserve strength to allow it to be pulled through the cut.

DISCUSSION

(Oliver Bowles presiding)

J. R. Thoenen, Washington, D. C.—Can Mr. Weigel give any idea regarding the overhead on the cost, above the nine cents mentioned?

W. M. Weigel.—That would be difficult. In the particular case described the overhead would be high, because the quarry was being opened, and the saw described was the only wire saw there. There would be the general foreman in charge, and whatever time the man that is backing the proposition might figure in as his overhead costs and the interest of his investment, and things like that. With three or four saws, the overhead would be divided between them.

J. R. Thoenen.—Do you think it would double the cost?

W. M. Weigel.—In this case it would, probably, because it is the preliminary development work, although they are shipping marble now.

J. R. Thoenen.—Even if it did double the cost, it would be cheaper than channeling.

W. M. Weigel.—I am not familiar with that, but probably it would.

S. A. Ionides, Denver, Colo.—Was there any difficulty in removing the blocks with the very narrow cuts made by that saw? How are they removed?

W. M. Weigel.—There was no difficulty. After the cut was made a series of holes was drilled underneath the cut, then the block was rolled over with a derrick.

S. A. Ionides.—Your photograph shows the channel cut, and that is the one I had particularly in mind.

W. M. Weigel.—That is the point where the ridge was broken up by fissures, so a tapered cut was made there, which in a 60-ft. cut was probably about 3 ft. narrower at one end than at the other. Those blocks were pulled out endwise, but after that first cut was out the blocks were simply rolled over.

Member.—Does Mr. Weigel know anything about the character of the sand and its effectiveness on the use of the wire saw? He speaks of the use of a round sand. Is it possible that an angular sand would be more satisfactory?

Do the cuts ever close up, as sometimes in the slate quarries, because the block slides?

W. M. Weigel.—The sand used was a local sand out of the White River, and as I say, it is derived almost entirely from St. Peter sandstone, which has a very rounded grain. This was easily obtained, and was delivered on the job at about one dollar per ton. A test will be made, however, with sand having angular grains.

In this case there was no tendency for the cuts to close; they were entirely open on one side. The stone was horizontally bedded and there was no tendency for it to slip. In fact, one wire broke and the next wire was worked right down through the cut from the top, after the old wire was pulled out.

O. Bowles, Washington, D. C.—I am much interested in this paper because there has been a question in the minds of some operators as to whether the saw would be successful in quarrying marble. Two marble companies that I know of have tried it out rather fully and neither one can get more than 2 sq. ft. an hour as a cutting rate, while this company in Arkansas has attained a rate of about 9½ sq. ft.; and in the slate district the average is around 25 sq. ft. per hour, with a maximum rate of 34 sq. ft. It seems remarkable that there should be such a difference in the cutting rate between marble and slate, and the rate attained by this company in Arkansas, which is apparently a profitable rate, makes it appear a little more promising than any information we have had previously.

H. MACMILLAN, Wilkes-Barre, Pa.—Do you not believe that a great deal of variation results from the use of different sands? I was in the Indiana limestone district where secondary or trimming cuts are made with the wire saw, and some of the companies there, after trying various local sands, have standardized on the flinty material obtained from the "chat piles" of the lead-zinc mines of the Tri-state District. I do not know the speed of cutting obtained, because that is difficult to estimate where trimming cuts are made.

O. BOWLES.—In the slate district there has been considerable experimentation with different sands—sand from New Jersey and Pennsylvania, sea sand and crushed quartz—and as nearly as I can determine there is not a very great difference.

H. MACMILLAN.—Of course the cutting effect will depend upon the difference in hardness of the wire, the quartz and the rock to be cut. If the quartz sand is much harder than either of the other materials, and of course it is, it would not itself be worn at all in the process of cutting. Therefore, it would appear that sharp, angular particles might give better results. I wonder if some attention might not well be given to the sharpness of the sand.

J. R. THOENEN.—When we started the first saw in the slate district we did some experimenting with rounded and angular sands, and really the results were better with the rounded than with the angular. Whether or not that was due to the size of the sand—there were several different sizes—I do not know, but my opinion is that any sand, whether rounded or angular when it starts into the cut, will be angular when it comes out. The saw breaks up the round pieces and makes angular pieces of them. The sand coming from the cut is always, of course, very fine.

Another thing, if the sand is a little too coarse it delays progress.

W. M. WEIGEL.—I do not believe there is much difference between rounded and angular sand, so long as there are no particles so coarse that they hold the saw up from the cut, and sand that is too fine would not be effective, of course. As long as it is all approximately the same size, it does not make much difference whether you start with round or angular pieces.

Barite in California

By Walter W. Bradley,* San Francisco, Calif.

(San Francisco Meeting, October, 1929)

Barite, or barytes as it is sometimes called, belongs to one of the lesser groups of nonmetallic minerals, of which 15 to 20 varieties are mined in California in amounts varying in value from a few hundred to almost half a million dollars annually.

Commercial production of barite in California, according to the records of the State Mining Bureau (now Division of Mines), began in the year 1910 with shipments totaling 860 tons, valued at $5640 f.o.b. rail shipping point. With the exception of the years 1924 and 1925, when no shipments were made, the output has varied, reaching a maximum of 17,993 tons, worth $90,617, in 1927. The 1928 output was 13,406 tons, worth $55,888 f.o.b. rail shipping point, bringing the state total to the end of 1928 to 59,972 tons, valued at $313,930.

Consumption of Barite in United States

Crude barite sold or used by producers in the United States for the year 1927 amounted to 248,219 tons, valued at $1,594,423, according to the U. S. Bureau of Mines.[1] In addition, there was imported, principally from Germany, a total of 70,274 tons, valued at $253,284. Manufactured products from these tonnages combined were distributed in 1927 (the latest year for which figures are at present available) as follows: ground barite, 57,658 tons; lithopone, 174,083 tons; barium chemicals (including carbonate, 4,960 tons; chloride, 3,541 tons; binoxide and sulfate or *blanc fixe* combined, 13,354 tons). Missouri accounts for nearly 50 per cent of the domestic crude barite, followed by Georgia and Tennessee, with lesser amounts from Arizona, California, Nevada, South Carolina, and Virginia.

Occurrence and Uses of Barite[2]

Barite, also called "heavy spar" and "tiff," is theoretically composed of 65.7 per cent of barium oxide (BaO) and 34.3 per cent of sulfur

* State Mineralogist of California.

[1] *Mineral Resources of the United States in* 1927, U. S. Bur. Mines. Preliminary Summary (Aug., 1929) A18.

[2] Occurrence and uses of barite have been described in detail. Some of the publications covering this subject are:

J. M. Hill: Barytes and Strontium. *Mineral Resources of the United States* 1915. U. S. Geol. Survey (1916). (Continued on next page.)

trioxide (SO_3) and has the chemical formula, $BaSO_4$. The specific gravity ranges from 4.3 to 4.6, and its hardness varies from 2.5 to 3.5. In the crystal form it is commonly found as an aggregate of straight or slightly curved cleavage plates, but it also occurs in granular, fibrous and earthy masses and in the form of stalactites. It is readily distinguished from calcite by its greater weight and the fact that it will not effervesce with acid. Commercial crude barites should carry more than 93 per cent barium sulfate, and the better grades of domestic barites carry at least 95 per cent $BaSO_4$; although in the case of lithopone manufacture lower grade material can be utilized without serious interference.

Barite occurs as a gangue mineral in many metalliferous deposits in the United States, but thus far apparently has but rarely been saved as a commercial product when treating the ores for their metal contents. One such utilization in the case of a Californian quicksilver ore is noted later in this paper.

As shown by the production data given in a preceding paragraph, more than two-thirds of the 1927 tonnage of barium products in the United States was lithopone. Barite occupies an important position in the list of mineral fillers used in the preparation of various manufactured articles and is used in the preparation of barium chemicals. One of the most important of the uses of ground barite, because of its high specific gravity, is in oil-well drilling mud.[3] Barite so used is ground so that 98 per cent will pass a 200 to 300-mesh screen. No standard specifications or tests are in use for either crude or ground barite.[4]

After several years' experimentation a successful process has been developed[5] by the Great Western Sugar Refining Co., Johnstown, Colo.,

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G. W. Stose: Barytes. *Mineral Resources of the United States*, 1918. Pt. II. U. S. Geol. Survey (1920).

Barites, Barium Chemical and Lithopone Industries, including Costs of Production. Tariff Information Series 18. U. S. Tariff Commission (1920).

H. S. Spence: Barium and Strontium in Canada. Canadian Dept. of Mines, Mines Branch *Pub.* 570 (1922).

Barytes Deposits of Georgia. Geol. Survey of Georgia, *Bull.* 36 (1920).

W. M. Weigel: Size and Character of Grains of Nonmetallic Mineral Fillers. U. S. Bur. Mines *Tech. Paper* 296 (1924).

W. M. Weigel: The Barite Industry in Missouri. *Trans.* A. I. M. E. (1929) 256.

R. B. Ladoo: Non-metallic Minerals. New York, 1925. McGraw-Hill Book Co.

C. C. Thoms: Use of Heavy Minerals in Rotary Drilling Mud in the Ventura Field. California State Oil and Gas Supervisor, 12th Ann. Rept. Calif. State Min. Bur. (1926).

Domestic Barium in Place of Imported Strontium. *Eng. & Min. Jnl.* (1929) **128**, 201. See also additional references later in this paper.

[3] C. C. Thoms: *Op. cit.*, 5.

[4] R. B. Ladoo: *Op. cit.*, 76.

[5] *Eng. & Min. Jnl., Loc. cit.*

using the hydroxide of barium instead of that of strontium, which is so largely used in Europe. A saccharate is formed by the union of the salt with the uncrystallizable sugar in the molasses, which is decomposed by carbon dioxide and liberates the sugar.

Californian Deposits and Plants

Barite is a common gangue mineral in vein deposits in California especially with galena, and therefore is prominent in the silver-lead districts. Some of the most important deposits are as follows:

Alameda County

There are no deposits of barite in this county but for some years there has been a plant at Melrose, in the southern limits of Oakland, engaged in the preparation of barium compounds. Originally built by the Barbour Chemical Co. about 1912, it treated barite and witherite from the deposit at El Portal in Mariposa County. Later it was operated by Lewis, Gilman and Moore, who began the manufacture of lithopone in 1920. The plant is now owned and operated by the Chemical and Pigments Co., Inc., principally in the preparation of lithopone, with a capacity up to 25 tons per day. Some acid-treated, ground barite is also prepared.

The company owns the Democrat mine in Nevada County, which is at present its principal source of barite. The mineral from the Democrat carries 88 to 90 per cent $BaSO_4$. In part it is almost black, due to carbonaceous matter, but this is an advantage rather than a detriment, as the ore is first mixed with 25 per cent of petroleum coke; it is then crushed to ¼ in., dried, and calcined to BaS, in which reduction the carbon assists. The sulfide is dissolved in water, then the iron and other impurities are precipitated out and filtered off. The zinc used is "zinc dross" from Anaconda, Mont., carrying 88 per cent Zn and some oxide, with cadmium, iron and other impurities. These are dissolved in sulfuric acid, and the impurities are removed by precipitation and filtration.

The zinc sulfate solution is run into the tank containing the barium sulfide solution and the mixed $BaSO_4$ and ZnS precipitate (lithopone) results; filtered, this becomes the "green cake," which is dried at from 500° to 700° C., according to the degree of oil absorption quality desired in the final product. This dried cake is then ground wet and dried in a special drier on a traveling screen, following which it is ground in a Raymond mill to -320 mesh and sacked in paper bags. A fractional percentage of Prussian blue is added to improve the whiteness.

Humboldt County

On Liscom Hill, 8 miles northeast of Arcata, a number of veins of white, crystalline barite have been noted, showing from a few inches to a

foot in width, and standing nearly vertical. Boulders of barite near by indicate that development work might uncover a workable deposit of the mineral.

Inyo County

Massive barite is recorded[6] as occurring in several localities in this county, and in veins in the Alabama hills. Shipments were made during the World War period from a deposit near Laws. At present, Joseph Smith, of Laws, owns two claims in Gunter Canyon on the west flank of the White Mountains, 6 miles northeast of Laws on the California and Nevada R. R. A series of parallel veins of barite 2 to 8 ft. wide occur in Cambrian schists and slates, and trend N.30°W., with dip 60° E. The barite is white, reported to carry 94 per cent $BaSO_4$, and a considerable tonnage is exposed on each side of the canyon.[7]

Los Angeles County

A deposit of white barite said to carry 85 to 87 per cent $BaSO_4$ occurs on the west side of San Dimas Canyon, 8 miles northwest of San Dimas. The width ranges from 6 to 8 ft., and the outcrop is visible for a length of 50 ft. There are also several small lenses on the ridge above. Some material was shipped from here several years ago but the property is not at present worked.

Mariposa County

The barite deposit on the north side of the Merced River, a mile west of El Portal, was the first, and for several years the only barite mined commercially in California. It likewise has the distinction of being the only deposit in the United States from which commercial shipments of the carbonate, witherite, have been made. The property was recently purchased by the National Pigment Co. from the Yosemite Barium Co., and was formerly worked at different periods by El Portal Mining Co., the Barbour Chemical Co. and Western Rock Products Co.

The river canyon cuts directly across the barite lode, as do also the Yosemite Valley R. R. and the new Yosemite state highway, the last-named being on the south side of the river. In March, 1927, development of the veins south of the river was begun and the present output is coming from these new workings, which comprise two adit levels at 80 ft. and 260 ft. respectively below the surface. The barite is mined by drifting and stoping. The vein width varies, showing a series of lenses,

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[6] A. S. Eakle: Minerals of California. Calif. State Min. Bur. *Bull.* 91 (1923) 262.

[7] W. B. Tucker: Mineral Resources of Inyo County. Rept. XXII of State Mineralogist of California (1926) 513.

average 14 ft., with a maximum of 35 ft. One of these lenses is estimated[8] to contain several hundred thousand tons of barite, averaging 94 per cent. $BaSO_4$. The company's holdings cover 500 acres. An aerial tramway carries the ore across the river to bunkers on the railroad.

W. D. Egenhoff, of San Francisco, has a group of three claims on a deposit discovered in 1917, 5 miles by trail from Jerseydale or 6 miles in an air line south from the nearest point on the Yosemite Valley R. R. Barite occurs in ledge form, between limestone hanging wall and quartzite footwall. It has been traced for a length of at least 4500 ft., and prospected by several crosscuts, but its size is not yet fully determined. Transportation is the principal difficulty.

Monterey County

On the slopes of Gabilan (or Fremont) Peak, along the line between Monterey and San Benito counties, there were a number of lenses of barite from which a considerable tonnage was shipped during the years 1916 to 1920 inclusive. The lenses were associated with silicified limestone and with dolomite. The barite mined was white and especially high grade, analyses showing 98 to 99 per cent $BaSO_4$. The lenses were apparently small, as there have been no further shipments. The material was hauled by motor truck to the railroad at San Juan Bautista, a distance of 9 miles.

Nevada County

The Democrat Barytes mine, 7 miles by road from Dutch Flat, is now owned and operated by the Chemical and Pigments Co., Inc. This deposit was first worked in 1919 by Bear Barytes Co. and in 1920 it was taken over by the Metals and Chemicals Extraction Co., which operated it for three years. The present owners began shipments in 1926.

The outcrop of the deposit, which has been mostly removed by mining, was in the shape of a dome or knob on the point of a steep ridge overlooking Bear River.[9] Of irregular shape, it varied in width from 15 to 24 ft. at the surface and had a length of 250 ft. rising about 60 ft. at the highest point above the level of an old ditch.

The ore of commercial grade, showing in part up to 98 per cent $BaSO_4$, passes into an impure barium-aluminum silicate rock on the north, and on the hanging-wall side there is no regular contact between ore and rock. The barite varies in color from pink or gray to black

[8] C. McK. Laizure: Mineral Resources of Mariposa County. Rept. XXIV of State Mineralogist of California (1928) 145.

[9] C. A. Logan: Unpublished report, California State Division of Mines (1929).

(due to carbonaceous matter), and is nearly all more or less "off color" when ground, requiring chemical treatment.

Mining for several seasons was by open cut, and later by a crosscut adit (100-ft. level). At present, ore is mined through a series of five raises put up from the 30-ft. level into the bottom of the old cut. From this level it is dropped to the 100-ft. level and trammed to the loading bin at the tram terminal, thence hauled 6 miles by 5-ton trucks to the railroad at Alta. For transporting the ore across the steep canyon of Bear River, there is a gravity aerial tram 1800 ft. long with 2-ton bucket. The ore has been mined for 250 ft. in length on the surface, and the 30-ft. level is 200 ft. long. At present 11 men are employed and shipments are at the rate of 100 to 150 tons per day.

Orange County

On Red Hill, 2 miles east of Tustin, cinnabar occurs in Tertiary sandstone with a barite gangue. On the north side of the hill, there are a number of small veins. Material from these was treated in retorts in 1927 for the extraction of the quicksilver. The calcined residue, barite, was sold to oil companies in one of the near-by oil fields for heavy-mud use in oil-well drilling operations.

San Benito County

See under Monterey County

San Bernardino County

Barite was common as a gangue in the silver districts of Calico and Barstow, and at the Imperial mine. It occurs 6 miles north of Barstow and a deposit is reported near Ludlow.

Santa Barbara County

One of the largest known deposits in the state, according to the records, is on the North Fork of La Brea Creek, 15 miles northeast of Sisquoc on the Pacific Coast R. R. A vein of white barite, with only slight iron stains, is exposed for several hundred feet along the top of the ridge; it occurs in a tough gray sandstone, striking east with the trend of the ridge and dipping steeply north. At the outcrop, the vein shows 25 ft. width, and analyses are stated to show 97 per cent $BaSO_4$. The owner is Bean Laughlin of Santa Maria, who has recently leased to Los Angeles interests. A road has been built to the deposit and ore is being hauled by truck to Los Angeles. It is reported that at least a part of the material is being utilized in oil-well mudding operations.

Shasta County

Barite has been shipped by H. C. Austin from a deposit near Copper City, but his quarry is idle at present. C. A. Packwood and Alonzo Luce, of Willows, have a group of two claims on a 4-ft. vein of barite, 12 miles north of Montgomery Creek and 44 miles by road from Redding. It is reported that the vein outcrops at intervals for 3000 ft. The Chemical and Pigments Co. Inc., made some shipments to its Melrose plant, from the Loftus group on Tom Neal Creek, 7 miles from Castella, but at present this is idle. Two egg-shaped deposits are said to have been found in limestone.

Mike Malone, of Platina, has two claims on a prospect containing witherite, on Beegum Creek, 41 miles from either Red Bluff or Redding. These are undeveloped.

Stanislaus County

The D-V-O Products, Inc., of which C. E. Gilman is president, has a plant at Modesto manufacturing a number of chemical products among which are included some of the barium compounds. Barite from the deposit at El Portal, Mariposa County, is used.

Summary

Barite resources of California are ample to supply the present market demands of the Pacific Coast, both for ground barite and for lithopone; and apparently they are ample for any considerable increase that may occur as the population and coast industries advance. Previous to 1920, when the first plant in California for preparation of lithopone began operations, it was not possible to dispose of anything but a high-grade, white, dry-grinding crude; but now with two plants for acid-treating available, there is a market for off-color grades of the crude mineral. With deep drilling in areas of high gas pressure, the demand for barite in oil-well mudding appears to be increasing in certain Californian oil fields.

Hydration Factors in Gypsum Deposits of the Maritime Provinces

By H. B. Bailey, Fredericton, N. B.

(New York Meeting, February, 1930)

Since the gypsum deposits of Nova Scotia have been operated on a large-tonnage basis, it has become increasingly necessary that more study be given to the geological relation of gypsum to anhydrite. It was while attempting to work out some definite system of quarry operation for several operators that the factors relating to hydration as given in this paper became apparent.

For convenience the words "the formation" are used to mean anhydrite and gypsum taken together as one bed geologically and forming a distinct unit from the rocks below or above. Also, the word "block" is used geologically to define a portion of the formation that is cut apart and separated by erosion from other portions.

The formation is part of the Lower Carboniferous, the remainder of which is made up of red shales and shaly sandstones, coarse red conglomerates and limestones. Overlying the Lower Carboniferous are the even-bedded gray sandstones of the coal measures. To a large extent these sandstones have been eroded from above the formation and the usual cover is clay and unconsolidated sediments.

The position of the limestone member which immediately underlies the formation usually furnishes the key to the general structure. Being harder, and subject to less erosion, it has a wider spread than the formation, and measures for dip read on it have a value which can not be applied to dips taken on gypsum or anhydrite. This limestone usually is not more than 25 ft. deep, but it has a remarkable tenacity—one would almost say elasticity—under strain of earth movements. In many cases it stands as a barrier to the erosion which otherwise would have destroyed completely the remaining block of the formation. No bedded limestone is found within the formation except in narrow seams close to the bottom and in short irregular masses in highly disturbed strata where it has been caught up by fault action.

This paper is not concerned with the general theories of original deposition and no opinion is expressed as to whether gypsum or anhydrite was first deposited from evaporating sea water. For purposes of study, and as an observed fact, except in the few cases where there has been extreme disturbance, a block of the formation is regarded as a block of anhydrite which has become partly hydrated into gypsum.

The vast size of the original basins of the Acadian deposits, as shown by the scattered fragments that remain, and the great depth and purity of the mass, do not fit in any satisfactory manner into the theories of original deposition which have been advanced. As they exist today the deposits are remnants only and in accounting for present conditions it is well to remember the great bulk of the original bed and the extreme degree of erosion to which it has been subjected during thousands of years. Great structural movements within the formation itself are often completely unexplainable in the relatively small amount of the remaining remnant.

The thickness of the formation in its deepest parts is still a matter of conjecture and no drill hole, so far as the author is aware, has been put down through the anhydrite into the strata below. At Hillsborough, N. B., the best section is afforded for estimation of depth, as the overlying conglomerate conforming to the formation can be seen readily. The depth indicated here is between 400 and 500 ft., and the writer has found no evidence of greater depth at other Acadian deposits. On the contrary, other Acadian deposits are somewhat thinner.

Hydration is the chemical action of water, which changes anhydrite to gypsum, but geologically the aspect to be studied comprises the physical conditions under which the water is applied to the anhydrite surface. Water pressure, temperature and time are obvious factors in this action, and of these three time is the main factor—ages of geological time. This paper hopes to show that there are less obvious factors of great importance, such as rock pressure and the medium through which the water is applied.

The words "top hydration," "bottom hydration," "side hydration,' and "interior hydration" are terms developed by the writer, to denote the position from which the water is applied.

Top Hydration

Top hydration is the action by which water is applied to the upper surface of the anhydrite and works downward. The result might well be called "normal gypsum" because, being evenly formed from the upper portion of the anhydrite, it is where it would naturally be expected. The anhydrite under this or any large area of anhydrite from which the top-hydration gypsum has been removed is "normal anhydrite."

Bottom Hydration

Bottom hydration has not been noted definitely in Acadian deposits but it is probable that the color of the "blue beds" at Windsor is due to manganese impregnation by capillary action from the limestone below.

It applies only on the lower and thin edge of a block where the pressure of the overlying strata is low.

In Blaine County, Oklahoma, bottom hydration is an important factor. It is due to the fact that the formation rests on soft red beds, on which the ground water spreads uniformly and is drawn upward by capillary action. The effect of this is to give two layers of gypsum with an unbroken core of anhydrite between, which is found over a wide area in that country.

Side Hydration

Side hydration describes what the writer believes has been, in its effect, the most puzzling factor to the investigators and operators of this district. It is the action of water as a hydration agent against the side of an anhydrite block rather than downwards from the top.

Interior Hydration

Interior hydration is a minor factor as compared to top or side hydration. It applies to hydration by slow-flowing underground streams and ponds within the formation. A good example of this action may be seen at the so-called underground lake in the Demoiselle Creek area of Albert County, New Brunswick. This is clearly a cavern due to hydration and solution contained in a solid block of anhydrite.

Overburden and its Relation to Hydration

It may be said that as a matter of economic interest the overburden will run from zero, or no cover, to 250 ft., because at greater depths than this the formation is likely to be entirely anhydrite and any gypsum that might be found would not be available for commercial operation. The overburden may consist of salt or fresh water, organic matter or forest cover; soft clays; glacial boulders or boulder clay; gravels, sedimentary rocks such as red conglomerate or gray sandstone.

All of these, from their physical and perhaps also their chemical nature, have an effect on the rate and amount of hydration; therefore it is not only the depth of cover but the kind of cover that indicates what is likely to be found beneath.

No Cover Condition

Without cover, the anhydrite is washed by rains, and rain water is pure and mobile water. There is little time for surface hydration, for as fast as the gypsum forms it is washed away. If the water finds cracks and penetrates into the interior of the anhydrite it becomes in time a more active hydration agent. It tends to gather into certain zones and finally

to have a slow percolating movement along lines of least resistance to the lowest point of outlet. In this way interior-hydration troughs are formed below the normal anhydrite level. Each particle of gypsum is greater in size than the former particle of anhydrite. The swelling pressure of

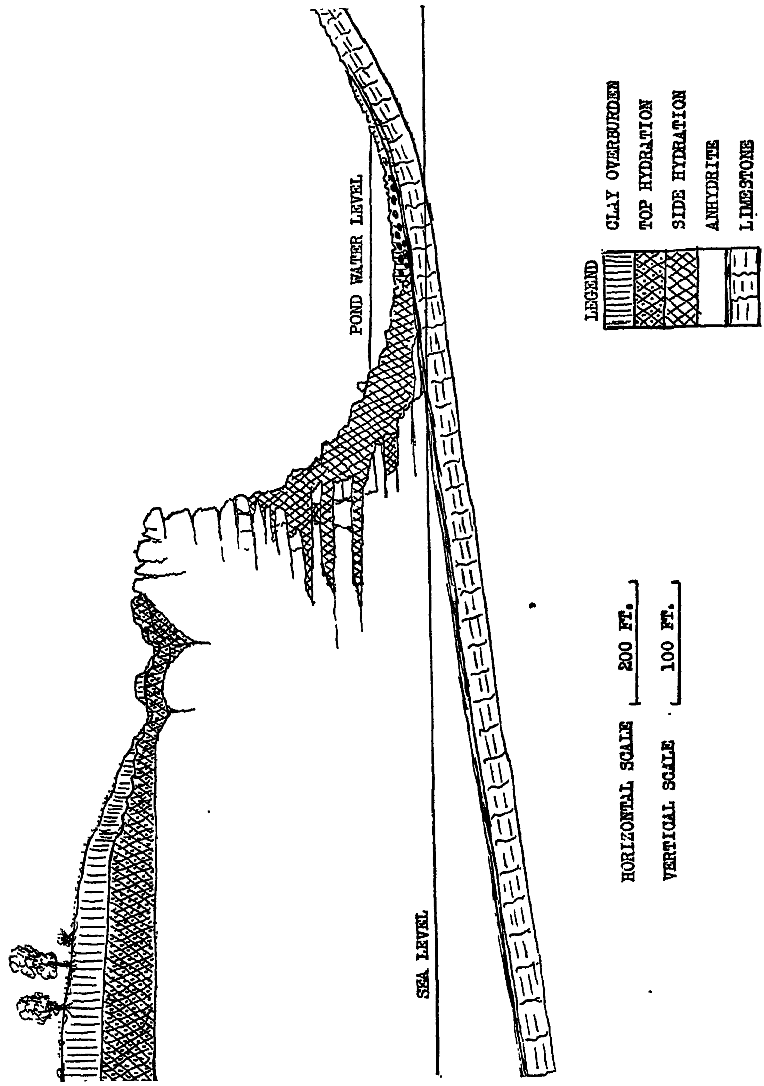

FIG. 1.—VERTICAL SECTION ILLUSTRATING TOP AND SIDE HYDRATION.

this action tends to form further cracks until the portion attacked is a ramification of gypsum veinlets in an anhydrite groundwork. It can be seen that a continuation of this action will lead to complete hydration.

Level or flat anhydrite ground, subject only to the action of rain, will develop a uniform size system of pot holes and these will be small

in diameter and with nearly vertical walls. This action would also lead to a surface enrichment of insoluble impurities. If the anhydrite contains small amounts of clay and iron oxide the solution of this rock will build up the impurities in the cracks, in the open-textured rock and bordering the pot holes. This accounts in some degree for the variation in surface color.

Water Overburden

Water overburden divides itself into that of fresh and salt water. In the Bay of Fundy district the salt water is tidal where it touches the formation. The bordering cliffs are mostly anhydrite, which appears to be casehardened by the action of the salt and to strongly resist erosion. Where gypsum is found it is usually highly discolored and filled with marl seams. The anhydrite condition can be well seen along the St. Croix River near Windsor, N. S., and at Pink Rock near Dorchester, N. B., there is a gypsum shore washed by the tide on which the rock shows a beautiful banding of bright red.

Fresh water over anhydrite in ponds would appear to hydrate the whole bottom uniformly as well as to form gypsum by side hydration. Near Windsor, a solid square lump of anhydrite 1 ft. thick, or more, will show gyspum on all sides for about ¼ in. in depth if the block has remained undisturbed in an old quarry pond for 50 years or so. This evidence, however, is not conclusive and no definite rate of hydration has been worked out.

Organic Matter or Forest Cover

Leaf mold does not accumulate in sufficient amount to have much action induced by it, and the influence, if any, of the weak acid solutions formed from it are unknown to the writer. Except for discoloration, the influence of this cover would be favorable as far as it goes. Tree growth, however, is important in preventing gypsum from being washed away when formed. By its interlocking roots it tends to retard the run-off of surface water and the gully-forming action these waters have, and it also tends to keep the surface moist, which helps to promote hydration.

Soft Clay Overburden

Soft clay is the usual overburden of normal top-hydration gypsum, and there is no better cover to promote hydration by what might be called the action of a wet blanket. The best possible clay overburden is unstratified—all of one kind and all of one color or texture from top to bottom. The more plastic the clay and the brighter the red color, with conditions of uniform texture, the better conditions will be below it. The iron oxide in the clay appears to act like a poultice and repeated experience has shown the finest white rock under the reddest of clay.

The writer's opinion has been that hydration under such clays was largely due to osmotic action. If such a conclusion could be definitely confirmed it would seem to offer a new angle from which the problem of the commercial conversion of anhydrite to gypsum might be attacked.

The best depth of cover would appear to be from 30 to 70 ft. of clay, and it will be noted that the top hydration frequently bears a close relation in depth to that of the cover.

Gravel

Gravel is not a common overburden on Maritime Province deposits but is an indication of severe stream erosion, where found.

Glacial Deposits

Glacial deposits consist of boulders and of boulder clays. The boulders are mostly of gray granite well rounded and from 2 to 5 ft. dia. The boulder clay forms low conical hills or ridges, with boulders of all sizes irregularly placed in a gravelly clay matrix. These can be seen bordering the Dominion Atlantic Railway back of Wentworth, N. S. As seen at Aspey Bay the glacial drift appears to have been caught in certain parts of the formation and to have built up ridges parallel to the obstruction. Such deposits appear to rest largely on anhydrite, so that their action in advancing hydration would seem to be slight.

Red Conglomerate

Red conglomerate as overburden was seen only at Hillsborough, N. B., but is of interest because it would seem that the deposits of red clay now overlying the gypsum must have been largely caused by the breakdown of this rock.

The jointing of this heavy-bedded stratum is such that water would reach the formation below along certain open fissure lines only. This would make irregular narrow zones of fairly deep hydration and form a patchwork of "soft" and "hard" rock within the limits and levels reached by the water. Locally gypsum workers refer to hydrated gypsum as "soft" rock; to partly hydrated and clear anhydrite as "hard" rock.

Cause and Effect of Hydration

For a study of the cause and effect of hydration, a supposed or ideal block representing average conditions found in these deposits will be considered.

As mentioned before, the block will be considered as at first entirely anhydrite resting on a limestone floor and the writer will attempt to give an outline of its evolution to its present-day maturity. Top hydration will start when the overlying rocks are largely removed by erosion or broken down into clay. Two principal structural changes commonly take place

The limestone floor becomes tilted up on the side away from the sea, usually with a dip not greater than 10°. On the edge of the sea, both parallel and also at right angles to the shore, a series of minor faults develop, caused by constantly changing levels in the bordering ocean floor. The evidence in most cases shows a subsiding coast line with the intrusion of the sea into old land valleys.

The high side of the block—away from the sea—is the point of most interest. If there is an even contour to the hill in the rear of the block there will be a fairly uniform flow of water down this slope and against the side of the formation.

If the block were small, surface streams would form and run around the ends; but with a mile or more of formation resting uniformly against the slope, the water tends to gather into a series of ponds or small lakes on the upper edge of the formation. Such lakes, and meadows marking their former position, are characteristic of these deposits.

It is clear that a pond not only supplies continuous contact water for hydration, but also that the water pressure, when there is any flow, is against the formation wall which holds it on the down-hill side. Such hydrating action forms gypsum in "bays" bordering the ponds and leaves "headlands" of anhydrite between them. This makes a scalloped or wavy outline to the formation wall as the ponds enlarge and reach lower levels. Erosion more easily follows hydration and when this happens the drainage is quickened. Still or slowly percolating water is a hydrating agent but rapidly flowing streams are erosional agents and cause a loss of previously formed gypsum.

There is a ratio of loss and gain between hydration and erosion, depending primarily on the angle of slope and area of the water shed. If the ponds remain, the water level will in any case gradually be lowered as the formation is dissolved away; and instead of the water resting against the top layers of the formation it will finally rest near the bottom, with the upper layers forming steep cliffs above.

When the upper portion is out of the water it will remain largely unchanged during a long period, while at and below the water level, hydrating action will continue. Water penetration in stratified beds is selective. In other words, hydration might advance in porous or seamy layers and not affect more massive ones above or below them.

When in these quarries 30 ft. or more of anhydrite is found with several seams of gypsum below—separated by narrow layers of anhydrite—there seems no reasonable explanation except that of the selective action of side hydration. The operator could expect in this case progressively more anhydrite as he worked away from the old pond limits.

The expectation held by many quarrymen that having low-level gypsum at one point they had only to break through a narrow anhydrite wall to find other low-level gypsum is not based on good reasoning, unless

it can be shown that some other hydration factor has operated behind the wall.

Any type of hydration will be promoted by ordinary fissures, cracks, bedding lines and joint planes but in some cases the fissure walls may be hardened by frictional pressure, or an impervious impurity may form a film over them. Both top and side hydration may be stopped by such impervious planes.

Taking a mass of anhydrite bounded by joint planes and subject to side hydration, the action would appear to be that if a crack were passed by the hydrating action the whole mass was likely to hydrate back to the next crack. It seems obvious to the writer that this and the selective actions before mentioned largely account for the puzzling checkerboard pattern found in side-hydration deposits.

Faults

Faults are much more common in the formation than is generally supposed but because of the softness of the rock they are hard to define. Both sides of the Bay of Fundy offer examples of this faulting. Lower Hillsborough, Walton and Avondale afford good study sections. The Bay has been throughout geological time an unstable trough. When its floor sinks or rises, it tends to fault the rocks bordering its shores. Most of these are minor faults. They are easy to note when found in the limestone but in anhydrite and gypsum there is much "drag" and recementation and the fault planes are seldom clearly defined.

The effect may be to shatter and toss the whole mass, and when this happens to anhydrite it naturally is much more subject to the hydration action and is likely to change largely to gypsum but to leave irregular masses of anhydrite where the action was weaker. With major faults, of course, the formation may be entirely lost to view or faulted out. There are also cases of diminishing faults where the effects will gradually decrease to zero away from the source of faulting.

In certain cases the vibrations from fault action have completely shattered the anhydrite and it has changed to gypsum by weathering without cover. Frequently lime and other impurities have entered by ascending waters. In a few rare cases small portions of other rocks are found thrown up and embedded in the formation.

Near Eastern Harbor on the Gulf Coast of Cape Breton, a mountainous table-land of pre-Cambrian rock skirts the seashore and in places is 1000 ft. high. The sedimentary rocks form a comparatively flat area between this table-land and the sea. The bright red beds of the Lower Carboniferous lie close to this mountain table-land wall and carry the gypsum formation. The contact zone between this Lower Carboniferous and the pre-Cambrian has been subjected to volcanic activity and in

places black basaltic rock has been intruded into the granitic wall. Usually the outer part of this table-land wall is step-faulted and the basalt has poured over it and filled the V-shaped valleys so formed. Since the close of the Lower Carboniferous age is known as a period of volcanic activity, it seems probable that this basaltic intrusion is of this period.

As at other gypsum deposits, anhydrite in horizontal position was probably the original primary mineral. Directly or indirectly as a result of this period of volcanic activity, the formation was thrown into a series of regular folds, which were again upthrust, compacted together by lateral pressure and faulted, to a repeating series standing nearly vertical. Only a small portion of the underlying red beds have been involved in the compacted structure, but the thin and flexible limestone mentioned as peculiarly the key member of the gypsum series structure has largely determined, by its strength and resistance, the final form of the whole mass. In such a structural mass practically complete conversion of anhydrite to gypsum through hydration is to be expected. This has taken place with little gypsum loss through solution, owing to the tight elastic synclinal key limestone at the base of the formation, in which the gypsum remains caught between the high retaining walls.

DISCUSSION

(Oliver Bowles presiding)

O. Bowles, Washington, D. C.—A good deal has been written at various times on theories of the hydration of anhydrite and gypsum, but this is the first really comprehensive paper that we have had by a practical man who has studied these conditions in the field and worked out problems of hydration of anhydrite into commercial bodies of gypsum.

H. J. Brown, Boston, Mass.—I have not seen Mr. Bailey's paper, and do not think that I am qualified to criticize it, but the theory is original, to say the least. There are some locations where considerable drilling has been done that do not seem to tally with this theory at all—particularly at St. Anne, on Cape Breton Island. I did some drilling there in 1913 and 1914, and went through successive beds of gypsum and anhydrite. The anhydrite was regular, hard, massive, and so was the gypsum, with no evidence of faulting. The strata were tilted, lying on the side of a hill, and I do not see how there could be any side hydration. With apparently no evidence of recent movement, with water coming down the rear end and at the sides, with gypsum on top and anhydrite below, I do not see how one can subscribe to a theory of this sort.

H. E. Brookby, Chicago Ill.—You have stated about the conditions of an ideal fault block. I imagine that what you have not done is to tie in all the geological data. For instance, you say that the formation was slightly tilted; it was probably tilted away from the shore.

H. J. Brown.—Certainly.

H. E. Brookby.—Probably there were fault blocks in a series along where the structure was broken, and perhaps a series of ponds. And the fact that there were

alternate layers of gypsum and anhydrite would indicate that your drill holes probably went through a local side-hydration zone. If you had drilled over a wide enough territory you might have found the limit of that zone. This theory developed by Mr. Bailey is designed to identify or isolate by a geological reconnaissance that part of a gypsum "formation" where the largest amount of hydrous gypsum occurs, and for this purpose it has demonstrated a considerable measure of usefulness. Mr. Bailey's theory is not intended to be a guide in operating a quarry of which the location has been fixed without reference to its local position in the formation, but rather to assist in finding a "location" in the formation where a quarry would encounter the most favorable conditions for hydrous gypsum.

H. J. Brown.—It would have to be a very large area. At Aspey Bay, the gypsum is right on top of the anhydrite, solid and over a pretty good-sized area. One would think that anhydrite would show some evidence of hydration, but that zone between the gypsum and the anhydrite is very clearly defined. You run from one to the other above water level. Of course, it cannot be said how long it has been above water level.

H. E. Brookby.—Aspey Bay has been studied within the past six months, and it conforms to the theories I have stated.

E. Moldenke, Windsor, N. S.—I have quarried from three to four million tons of gypsum during the past four or five years, and I have come to the conclusion that no theory is applicable. One day a quarry face is all gypsum and the next day the first blast may show all anhydrite behind it. It is much more complicated than Mr. Bailey's paper would indicate.

H. J. Brown.—If Mr. Bailey has evolved a theory that will hold water, he has given us something no one else has ever given, and he deserves great credit.

Scope of the Light-weight Aggregates Industry*

By H. Herbert Hughes,† Washington, D. C.

(New York Meeting, February, 1931)

The trend in modern building construction is definitely toward the use of weight-reducing materials. The basic advantage of lighter structural weight is obvious; reduction of dead load with retention of equivalent strength affords the possibility of increasing the live load, or if this is not desirable or necessary it makes feasible a reduction in size of structural steel members and corresponding savings in other phases of construction.

Among the weight-reducing materials and practices rapidly gaining favor are the use of light-weight patented insulating board to replace back-up tile or units, particularly in home construction; introduction of aluminum alloys to replace steel for many purposes; manufacture of porous brick, the use of which materially reduces the weight of masonry walls and partitions; and the use of self-rising vesicular concrete, diatomite and sawdust tile, porous gypsum, or gypsum tile for nonload-bearing partitions and walls. Even building accessories are being designed to save weight and space; one of the most striking examples of this is improved radiator construction.

Concrete technique is keeping pace with the rest of the industry. Burned shale aggregates now available in many sections of the country will make concrete weighing only 100 lb. per cubic foot, saving roughly 35 per cent. in weight and sacrificing none of the strength of a rock-sand mix. It may be more expensive, but actual experience has shown that the saving in dead load will effect a reduction in structural steel which in many instances will more than offset the increased cost of the aggregate. The vesicular nature of most light-weight aggregates gives the concrete especially good insulating, fireproofing and soundproofing properties; yet absorption is comparatively low. Strangely enough, however, the increased use of light-weight aggregates appears to have augmented the demand for ordinary aggregates in concrete construction rather than to have decreased it. Light-weight aggregates have widened the field for concrete structures and, since the producers do not attempt to recommend the use of their material except where weight or other special properties are important factors, impetus has been given to sales of other aggregates.

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* Published by permission of the Director, U. S. Bureau of Mines.

† Assistant Mineral Economist, Building Materials Section, U. S. Bureau of Mines.

The largest outlet at present for light-weight aggregates is in the manufacture of precast building units. All standard sizes of tile and blocks are made at most plants as well as special header, joist, chimney and steel sash blocks, lintels and brick. Odd sizes and shapes can be made to order. Most light-weight units can be nailed, sawed or channeled which, together with their light weight, strength, insulating qualities, fireproofing and soundproofing properties and economy, makes them popular in building construction.

Kinds of Aggregates

All light-weight aggregates fall into one of three divisions, depending upon their source: (1) Those which occur naturally, such as volcanic cinder, tuff, pumice and coal; (2) those formed as by-products in industrial processes, including cinders, slag and sawdust; and (3) those manufactured specifically for use as concrete aggregate. The third group includes, among other materials, such trade-marked products as Haydite, Pottsco, Cel-Seal, Lytag and Corlite.

The purpose of this paper is not to enter into a detailed discussion of the properties or relative advantages of these various aggregates, but rather to call attention to the rapid development of this comparatively new industry, and incidentally, with the help of the accompanying map (Fig. 1) to outline the development and the present production status of the better known products.

The light-weight aggregate producers, the State Geologists of the western states and the United States Geological Survey, have aided materially in the preparation of this report.

Naturally Occurring Aggregates

Volcanic Cinder, Tuff and Pumice

Volcanic cinder, tuff and pumice are the only common rocks light enough and strong enough to be classed as light-weight aggregates. Volcanic cinder, or scoria as it is sometimes called, is irregular, clinkerlike, vesicular lava, either thrown out of an eruptive volcano or formed by the breaking up of the crust of a lava flow caused by rapid cooling. Tuff is a consolidation into beds of volcanic ash, cinder and dust. Many tuff deposits are too fine to be of any value as coarse aggregate. Pumice is a cellular glassy lava sometimes likened to volcanic froth. Small quantities of all of these materials have been used locally for concrete aggregate.

The deposits of such rocks are confined to the Rocky Mountain and Pacific Coast States. Most of them are far removed from railroads and markets, and this inaccessibility renders them commercially valueless. Furthermore, lack of uniformity of the material throughout a deposit makes much of it unsatisfactory for concrete. During the search for

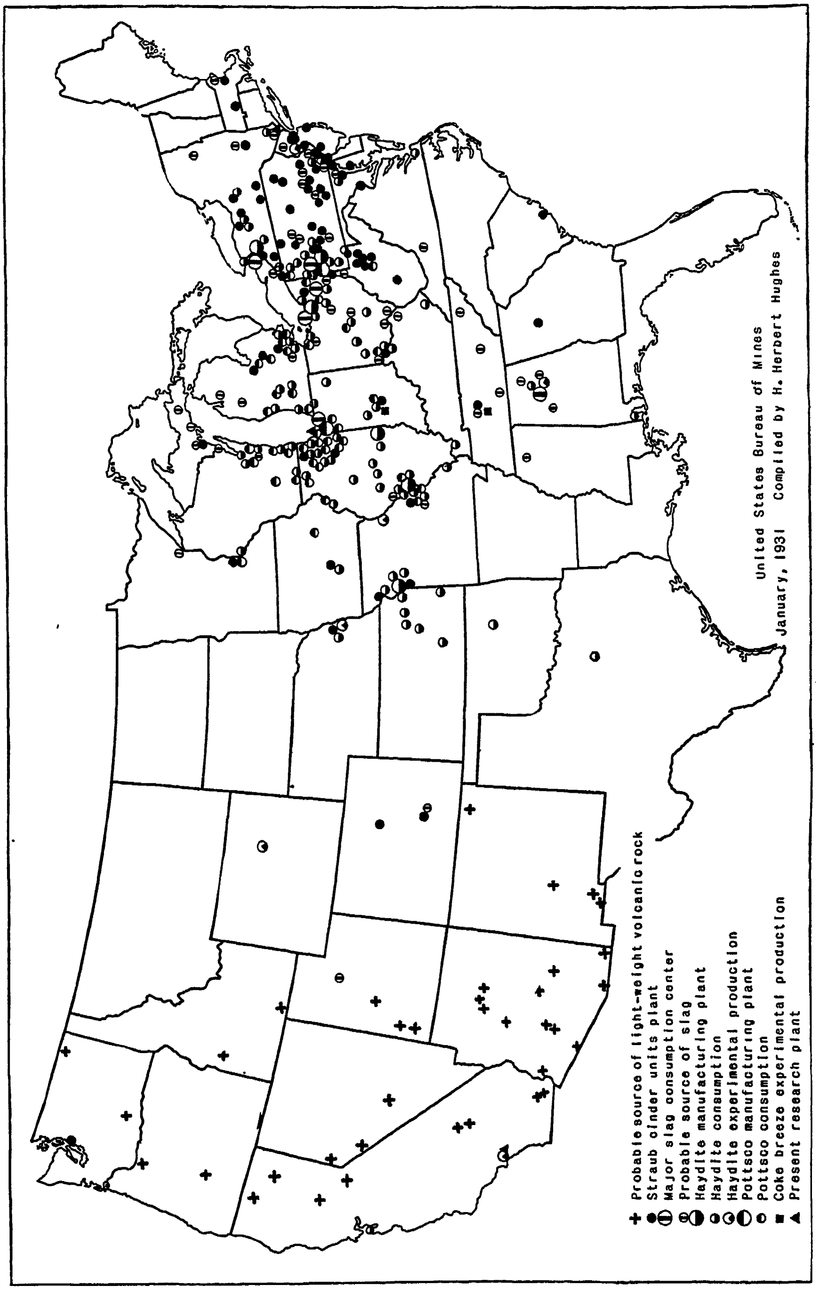

FIG. 1.—DISTRIBUTION OF LIGHT-WEIGHT AGGREGATES.

light-weight aggregate for use in concrete ships many deposits were investigated, some of the tests giving good results, but none of the material was actually used in ship construction because of its inaccessibility. It has since been used sparingly for light-weight concrete, interesting examples being the roof of the United States Army Crissy Field hangar, Presidio, San Francisco, and a bridge deck constructed by the United States Steel Corpn. in the State of Washington. The aggregate for both these jobs was a light pumiceous rock.

Arizona has probably the largest reserves of light-weight volcanic rock suitable for concrete aggregate. Although none of this material has been used to any extent as aggregate, several of the tuff deposits have been quarried as building stone. It has been used in the Arizona State Capitol, in one of the University of Arizona buildings and in numerous smaller structures. It is reported that supplies of unconsolidated cinders may be found northeast of Flagstaff and elsewhere in the state.

The search during the war for aggregates suitable for concrete ships disclosed two deposits of good material in New Mexico—one at Des Moines, Union County, and one at Cutter, Sierra County. Neither deposit was ever used, and no further development in New Mexico has been attempted.

California, because of its greater market possibilities, has been more active in the utilization of natural light-weight aggregates, but as yet their use has been confined to small local jobs. It is reported, however, that the possibility of large-scale production of natural light-weight volcanic rocks is now being carefully investigated, and that the preliminary survey of the situation appears to be favorable. Transportation costs no doubt will decide the practicability of the plan, because, unfortunately, the deposits of best material lie at a considerable distance from the consumption centers.

Pumiceous material from Millard County, Utah, has been used locally for light-weight aggregate with satisfactory results. Minor deposits of porous volcanic rocks are found in Nevada, Utah, Idaho, Oregon and Washington, but little is known concerning the material because many of the areas are practically inaccessible.

The locations shown on Fig. 1 indicate areas in which light-weight rocks have been reported or may be expected, not actual deposits of material suitable for concrete aggregate.

Coal

Anthracite coal has been used experimentally as concrete aggregate, the concrete having good strength and, strange as it may seem, fairly good fire resistance. Such use of coal, however, is uneconomical and at present of no practical value.

By-product Aggregates

Cinders

Steam-coal cinders, supplied principally as by-product material from electric power plants, large industrial concerns and well-operated railroads, are the most commonly used and the most widely distributed of the light-weight aggregate group. Cinders preferable for use as aggregate are a product of high-temperature combustion as differentiated from household ashes produced at a temperature too low to cause sufficient clinkering to give the material adequate strength to be used as aggregate. Producers of large amounts of cinders have learned that this one-time waste material is really an important commodity in many localities and, as a result, their combustion engineers are interested in maintaining uniformity of the product in order to secure the additional revenue.

Lack of uniformity is the principal objection to the use of cinders as aggregate in structural concrete. For the manufacture of precast building units, however, cinders have been entirely satisfactory, and that branch of the industry has enjoyed remarkable growth.

Cinders have been used as concrete aggregate in various proportions and mixtures ever since the introduction of portland cement. No statistics are available to show either the quantities of cinders used in structural concrete or the consumption centers, but it can be said with reasonable accuracy that cinders are used for aggregate wherever supplies are available. They reach their maximum importance in cities along the Atlantic seaboard, particularly in New York and Philadelphia, where cinder concrete is commonly used in many phases of building construction.

The development of cinder building units has had an entirely different history from cinder concrete, although here again cinders and various proportions of cinders with other aggregates or admixtures had been mixed with portland cement and cast into blocks for several years before the Straub patent.

Francis J. Straub, however, was the first person to visualize the commercial possibilities of large-scale production of cinder units. The patent granted to him in 1917 has been attacked in court several times but its validity has been sustained, partly because his conception has founded a great industry, whereas all previous operations had scarcely passed beyond the experimental stage. Furthermore, the simplicity of Straub's claim is diagnostic in itself. His block uses all the original mass of cinders and ashes, whereas his predecessors as well as his more recent would-be infringers used complex formulas and preparation methods.

Straub cinder blocks possess the characteristic properties of light-weight concrete units, and their widespread use is sufficient endorsement

of their quality. Production of cinder units grew from 18,000 manufactured by Straub in 1919 to 24,533,821 units produced by 77 licensees in 1926. The output since 1926 has been more nearly constant. At present more than 80 plants are licensed to manufacture cinder products and no doubt there are still a few small infringers whose operations have not yet been noticed by the controlling company.

Growth of the cinder units industry will be governed by increased output rather than geographical expansion because necessarily it will be confined to areas where coal is the principal fuel. The southern and southwestern states burn large quantities of gas and fuel oil, thereby eliminating the possibility of extensive manufacture of cinder units in those areas. The locations of the cinder-units plants on Fig. 1 roughly outline the region in which cinders suitable for light-weight aggregate are available. No attempt has been made to show the distribution of cinders for aggregate in light-weight structural concrete, but it may be inferred that such areas are represented with reasonable accuracy by the locations of plants making Straub units.

Slag

Slag is not strictly a light-weight aggregate and the members of the National Slag Association make no attempt to advertise it as such. It is included in this discussion, however, because the average weight of air-cooled slag concrete ranges from 130 to 140 lb. per cubic foot, which is 10 to 20 lb. lighter than rock-sand concrete, and because a process now in use produces a light-weight water-cooled granulated slag highly satisfactory as aggregate in precast building units. Water-cooled slag with emphasis on this special product will be described in the section devoted to specially prepared aggregates.

Blast-furnace slag is the nonmetallic residue formed as a by-product in iron manufacture. Its composition is rather complex; it consists principally of calcium, magnesium and aluminum silicates. Air-cooled slag breaks into harsh, angular fragments varying in color from all shades of gray to green gray and brown gray. It weighs from 65 to 100 lb. per cubic foot, the maximum in concrete aggregate sizes being about 85 lb. Concrete made with slag aggregate weighing less than 65 lb. per cubic foot has been unsatisfactory in most instances.

Only 4 per cent of all aggregates used in concrete is slag, but its utilization is well centered and slag aggregate is of major importance in some districts. Slag has been used in varying proportions as concrete aggregate ever since the advent of portland cement, and slag concrete has been used in all types of building, bridge, foundation, and even road construction during the past 50 years, exhaustive tests showing that its properties and its adaptability to most jobs are essentially the same as rock-sand concrete.

The production of slag is necessarily directly allied with iron and steel manufacture and prohibitive freight rates limit its distribution. The most important production as well as consumption centers of blast-furnace slag for concrete aggregate are Pittsburgh, Pa.; Buffalo, N. Y.; Youngstown and Cleveland, Ohio; Chicago, Ill., including Gary and Hammond, Ind.; and Birmingham, Ala. Large quantities of slag are used in New York and in Philadelphia where the contractors seem to regard it as light-weight aggregate. Other slag-producing districts of lesser importance are eastern Pennsylvania; Wheeling, W. Va.; northeastern New York; the northern peninsula of Michigan; and Pueblo, Colo. Every blast furnace, whether or not it is in blast at present, is a possible source of aggregate and on Fig. 1 two symbols have been used to show slag distribution; the large circles represent the principal production centers, whereas the smaller ones show the location of all furnaces regardless of whether the slag is now being used as aggregate.

Sawdust

Sawdust is practically unknown as light-weight aggregate in this country. In Europe, however, both sawdust concrete and sawdust-diatomite cement tile are used rather extensively because of their low cost and their light weight. Even the difficulty of swelling on wetting and shrinking on drying is reported to have been overcome by a secret process of treating the sawdust.

If the use of sawdust as concrete aggregate spreads to America its utilization will provide a further outlet for the immense quantities of by-product sawdust available in the big lumbering centers, particularly in the southern and the Pacific Coast states. Both of these regions are vast potential markets for light-weight concrete, for none of the present aggregates in commercial production has entered either area to any appreciable extent. The successful introduction of sawdust concrete should certainly be beneficial to both the lumbering and the building materials industries.

Manufactured Aggregates

Haydite

Haydite is a light-weight burned shale aggregate, developed and patented by the late Stephen J. Hayde. His original idea, conceived prior to 1905 and patented in that year, covered the use of burned clay, water and cement for making concrete. This process utilized waste material from ceramic plants, the fireproofing qualities of this early material being the principal consideration.

Hayde continued his experimental work and in 1913 produced the first sample of material comparable to the present product. The first

patent was granted to him in February, 1918, and shortly after this he offered it gratis to the Government for production of Haydite to be used in concrete ship construction. This patriotic move, unfortunately, resulted in extended litigation, but all the suits as well as subsequent minor litigations have finally been decided in favor of the Hayde patent. Furthermore, a reissue of the patent was granted in 1927.

Haydite is a vesicular, clinkerlike aggregate which, because of its semivitrified nature, has exceptional strength, considering its light weight and cellular structure. It is produced by burning in a rotary kiln a clay or shale, which retains its original chemical moisture content as it enters the kiln. Preheating at the charging end of the kiln vitrifies a thin layer on each particle, which prevents the gradual escape of gases during burning. Near the discharge end the material is subjected to high temperature and the resulting semifusion permits the sudden release of pent-up gases, causing each particle to expand into a porous clinker. After cooling and thorough wetting, the material is crushed and screened. One fine and two coarse sizes are produced. The average screen analysis of the sand-size Haydite is 17 per cent retained on 14 mesh, ranging to 12.5 per cent passing 100 mesh with a fineness modulus of 2.65. The coarse grade is composed of ¾-in. to 4-mesh particles and the intermediate grade of ½-in. to 4-mesh; their fineness moduli are 6.70 and 6.20, respectively. Absolute control of the process insures uniformity of the product.

The weight of Haydite varies from 1500 to 1600 lb. per cubic yard for the sand size to about 1200 lb. for the ¾-in. In computing shipping weight an additional allowance of 100 lb. must be made to cover moisture content.

Haydite aggregate is especially desirable for structural concrete for all purposes where weight and strength are important factors. The average weight of Haydite concrete is only 100 lb. per cubic foot, a decided reduction from concrete made with ordinary natural aggregates. The design of any building determines the particular parts of the construction where Haydite may be used advantageously. It is not recommended indiscriminately for use in all concrete work because its slightly higher cost may make such a practice uneconomical. Its intelligent use, however, will result in net savings in many types of construction because of its high strength and the marked reduction in dead load. Haydite concrete is especially adaptable to bridge floors, particularly in long spans where reduced dead load means appreciable savings in structural steel.

About one-half of the total Haydite production at the present time is used as aggregate in the manufacture of precast light-weight building units, which are highly satisfactory for all purposes where light-weight units can be utilized advantageously. The successful use of Haydite concrete and Haydite masonry units in hundreds of structures shows

conclusively that its development has definitely passed the experimental stage and that its growth may be expected to continue in the future.

Eight plants with a total of 14 kilns were manufacturing Haydite in the United States and Canada during 1930, as contrasted with one kiln in 1918 and only two in 1926. Production has grown from less than 25,000 cu. yd. in 1925 to about 200,000 cu. yd. in 1929, with 1930 yardage estimated to be higher.

The plants in the United States are at Kansas City, Mo.; East St. Louis and Danville, Ill.; Cleveland, Ohio; Pittsburgh, Pa.; and Buffalo, N. Y. Manufacturers of concrete products within reasonable shipping distance of each plant supply building units to near-by markets. The use of Haydite as aggregate in structural concrete, particularly for bridges, has greatly extended the territory in which it has been marketed. Aggregate from the Kansas City plant has been shipped to both the Atlantic and the Pacific coasts for use in bridge construction. The symbols on Fig. 1 show Haydite manufacturing plants; principal Haydite consumption localities, both as precast masonry units and poured concrete; and experimental plants, most of which were operated only during the war to manufacture Haydite for use in concrete ships. The significance of these plants, even though they are no longer productive, is that they indicate locations where shale, no doubt suitable for Haydite, is available.

Haydite is less dependent upon special raw materials than other aggregates of the light-weight group. Practically any shale or clay is satisfactory, although material containing some carbonaceous matter gives the best results. Judging from the present trend, Haydite appears to be moving toward Atlantic seaboard markets, but the manufacturers certainly will not overlook the large potential markets of the South and Southwest, where burned shale aggregates will be free from the competition of other members of the light-weight group which are excluded from those areas because of the lack of raw materials.

Pottsco

Water-cooled granulated slag has attracted attention as light-weight aggregate for several years, particularly for use in masonry building units. No extensive utilization of the material has ever been made, however, because of its soft friable nature and its low crushing strength. The H. H. Potts Co. has overcome these objectionable features by a special cooling process. The product, Pottsco, was first marketed in the Chicago district in August, 1928.

The Pottsco manufacturing process is completed within the steel mill, using slag from selected furnaces. Patents covering its manufacture are pending and no details regarding the process are yet available. Regulated temperature of the water during cooling appears to be the important feature.

Pottsco aggregate weighs about 1500 to 1600 lb. per cubic yard, a liberal moisture allowance bringing the shipping weight to 1800 lb. One commercial grade is produced, the size of the particles ranging approximately from 4 per cent. retained on 8 mesh to 98 per cent. retained on 100 mesh, giving a fineness modulus of 2.85. At present Pottsco is being used almost entirely for precast masonry units. Tests have been made, however, covering its use in poured concrete, particularly for floor sill and roofing, but the company has not actively promoted its sale for these purposes.

Pottsco masonry is recommended for all types of construction requiring back-up or partition units, its insulating properties being especially stressed. The aggregate has been available only since 1928, but its reception in the building trade shows that it occupies an important position in the light-weight field.

The plant supplying Pottsco to the Chicago district is at Indian Harbor, Ind., and from there the material is shipped to the concrete products plants that manufacture Pottsco units. Most of these plants lie within a 300-mile radius of Chicago. The maximum capacity of the Indian Harbor plant is 750 tons per day. A second plant for production of Pottsco was opened in Pittsburgh, Pa., in September, 1930. Its maximum output is 1000 tons daily, but distribution in this area is still in its infancy. Fig. 1 shows the location of these two plants as well as districts of appreciable consumption of Pottsco. Each one of these districts, with a few exceptions, also represents the location of a plant manufacturing Pottsco units.

Pottsco production has increased steadily since 1928 and it is reasonable to assume that the increase will continue. Ultimate Pottsco production, however, will be confined to those areas where suitable slag is available. Buffalo, Youngstown, Cleveland, and especially Birmingham, are logical locations for future plants although the slag produced in the minor iron and steel manufacturing districts also may be utilized. Expansion of Pottsco production, particularly for building units, will not only increase the field of light-weight aggregates but also will aid the iron and steel industry in profitable utilization of by-product slag.

Lytag

Lytag is the trade name of a light-weight aggregate now being produced experimentally in Chicago. It is a burned shale or clay product manufactured under patents which protect the process as well as the machinery used in its manufacture. Practically every shale, clay, and even sand or loam will show vesicular structure when sintered by the Lytag process, but easily fusible common shale and clay will give a more satisfactory product at a cheaper operating cost.

The sintering process for making Lytag is unique in the light-weight field. Shale crushed to 4-mesh fineness is mixed with a small proportion of granulated coal in a pug mill, an important feature of the process being the addition of moisture. It is then spread on grates over suction chambers and ignited by a flame applied for only about 30 sec. Combustion without flame continues downward, aided by down draft. The combustion process may be likened to smoking a pipe, a match lights it, suction keeps it ignited and the ashes correspond to the sinter that remains in the grate. The machine employed apparently corresponds, at least in principle, to the Dwight-Lloyd sintering machine extensively used for roasting and calcining ore.

The properties of the sinter are more dependent upon the process itself than upon the raw shale. The operator may vary the shale-coal ratio or the moisture content, he may change the speed of revolution or the depth of the charge, or he may add other constituents to the mix. All these variations will alter the nature of the product and this flexibility of operation is one of the principal advantages of the process. A peculiar feature is that the vesicular sinter shrinks in size rather than expands, as is characteristic of other burned shale aggregates.

No commercial production of Lytag has yet been attempted and no accurate information regarding properties and tests of the material has yet been published.

Cel-Seal

The only light-weight aggregate being produced on the Pacific Coast is Cel-Seal, a burned clay product manufactured in Los Angeles, but it also has scarcely passed the experimental stage as yet.

Cel-Seal is made by pugging a mixture of soil and clay and forcing it through a die. It is then broken or cut into pieces of various sizes, each one of which is covered with a thin coating of fine silica sand, the sand coating serving to keep the fragments from sticking together. After burning in a rotary kiln the resultant clinkered particles are screened to the desired sizes.

Assuming that Cel-Seal can be manufactured on a large scale at a reasonable cost, the location of the plant is well chosen, because lack of raw materials will prevent the light-weight aggregates other than burned shale products from being produced in the West Coast territory.

Corlite

Corlite is a light-weight aggregate produced by chemically treating anthracite ashes, adding a flux and sintering the mixture. A plant has been erected in New York City, but little has been published regarding the scope of the company's activities. Production still appears to be in the experimental stage.

Since production of Corlite will necessarily be confined to areas where consumption of anthracite is high, the New England and the North Atlantic seaboard states appear to be the logical territories for its manufacture and use. Fortunately for the Corlite Corporation these states, and particularly New York City, where the present plant is situated, constitute a vast market for light-weight aggregates.

Miscellaneous

Coke and coke breeze have been used experimentally as light-weight aggregates in precast building units. Extensive experimental work and even actual commercial production were carried on in Nashville, Tenn., and Indianapolis, Ind. Even though coke breeze is a fairly satisfactory light-weight aggregate, its use is uneconomical and there is little possibility that it will ever develop extensively.

Another interesting product, although not strictly a light-weight aggregate, is the naturally clinkered clay overlying vast beds of burned-out lignite and semibituminous coal in North and South Dakota. This material is used for road metal and railroad ballast, and locally it may be vesicular enough to have light-weight properties, but the supplies of such material are inadequate for any commercial production.

Summary

Fig. 1 strikingly reveals the segregation of light-weight aggregate production in well-defined areas. The most prominent example is the Chicago district, where Pottsco, Haydite, cinders and slag are in direct large-scale competition. The Chicago market, however, readily absorbs all these materials; no statistics regarding cinders and slag are available, but both Haydite and Pottsco are gradually increasing in volume of production.

The situation in the Pittsburgh district is rapidly assuming a similar appearance. Regardless of the many cinder units plants in western Pennsylvania, both Haydite and Pottsco are now being produced near Pittsburgh. Both of these operations are comparatively new but the preliminary reception given the aggregates by the Pittsburgh market indicates that their expansion is assured.

The Haydite plants in Buffalo and Cleveland are competing with cinders and slag, and Pottsco naturally may be expected to enter both districts. Cinders, slag, Pottsco and Haydite are all available in Detroit, although neither Haydite nor Pottsco is manufactured there. Haydite is manufactured in East St. Louis, Ill., and it as well as Pottsco, cinders and slag are available in the St. Louis area. Haydite is well established in the Kansas City district, competing only with cinder aggregate.

The vast markets for light-weight aggregates in eastern cities, especially Boston, New York, Philadelphia, Baltimore and Washington, are

being supplied entirely by cinders and slag, with the exception of small quantities of Haydite shipped from the Pittsburgh plant. Both Pottsco and Haydite, evidently attracted by the possibilities of finding an outlet for large-scale production of their aggregates, are expanding eastward. Pottsco production will be handicapped in some eastern localities because of the lack of suitable raw slag.

The southern, southwestern and Pacific Coast states must not be overlooked. Slag is available in the Birmingham district either for direct use or for Pottsco manufacture; but elsewhere in this extensive area burned shale aggregates could be manufactured with little competition from other members of the light-weight group. Either Haydite expansion or introduction of new burned shale products now in the experimental stage will no doubt extend the field of light-weight aggregates to include the principal cities in these sections of the country.

Light-weight aggregates unquestionably occupy an important position in building construction, and if past production records may be taken as a criterion for future predictions, any doubt regarding the continued success of the light-weight aggregates industry as a whole is dispelled.

DISCUSSION

(Oliver Bowles presiding)

F. A. Glass, Chicago, Ill. (written discussion).—The principles and the technique of the sintering process, and the continuous type of equipment widely used on a large scale in the metallurgical industry for desulfurizing and agglomerating small particles of ore and other metal-bearing particles preparatory to smelting them have been adapted and applied to the manufacture of light-weight vesicular aggregates of ample strength for all classes of concrete. Clay, shale, slate, schist and other earth substances containing hydrous silicates, or of an argillaceous nature, are deemed most suitable for such purpose, but anhydrous materials like blast-furnace slag, pure silica sand, burnt clay, ashes, etc., have yielded vesicular aggregates of fair quality. For convenience, all of these materials are termed clay, though they may lack plasticity, alumina, or hydrous silicates. Occasionally it is desirable to use anhydrous materials for flux; to facilitate the working of sticky clays; to improve the structure of the finished products; or to utilize what otherwise would be waste.

Structure, or the size and multiplicity of the vesicles and communicating passages in sintered aggregates, is of great importance, and fortunately most of the factors that influence it are within the control of the operator. Foremost is control of the water content of the raw charge of clay and fuel. In general, maximum bulkiness seems to be desirable, and is attained through water control, by gathering together the smaller particles of the charge into little clusters or pellets, and placing them loosely and in a freely permeable condition upon the grates of the sintering machine. Other important controllable factors are the fuel-clay ratio; the velocity and volume of air drawn through the charge in sintering; and the intensity of the ignition flame momentarily applied to the surface. The natural factors that have some influence on the structure of the finished products are mineralogical composition; size of the individual minerals; and the degree to which the minerals are united. Between very wide limits chemical composition seems of minor consequence; and those qualities of clays that ceramists

have found unsuited for use in their art either are not seriously objectionable or are beneficial. Certainly those clays that under heat-treating processes cannot be readily prevented from yielding products of vesicular structure are desirable for the purposes of those interested in aggregate manufacture. Likewise carbonates of lime and magnesia in sufficient quantities to cause the sudden fluxing of clays at about the temperature of incipient fusion are not a cause of concern to those desirous of conducting sintering operations with dispatch.

Fuels of very high ash content, and even carbonaceous shale, are suitable for aggregate manufacture because the incombustible solid contents are of the same nature as clay. All the incombustible solids from which aggregates are made fuse together, the ash content of the fuel becoming an integral and indistinguishable part of the sinter. Sulfur occurring as pyrite and marcasite burns or escapes as a fume, and the iron content, uniting with the silicates, acts as a beneficial flux. Refuse from the coal washeries of the anthracite regions yields excellent vesicular aggregates from even the leanest products, which usually have more than sufficient carbon for sintering, while the refuse of high carbon content, but just below the limit of marketability, yields good aggregates when diluted with clay deficient in combustible constituents. Even shales of light color often have nearly enough combustible components for sintering.

An unusual problem in aggregate manufacture arose in examining a gravel or calcareous boulder clay of the glacial drift in a population center where more suitable material was not available. The Dorr trommel classifier proved to be an effective and economical means of beneficiating such material; the rake product was perfectly clean and suitable for sand aggregate; the trommel oversize, with reasonable care, can be made free from clay and suitable for gravel aggregates; while the classifier overflow can be run into mud ponds and then reclaimed for use in sintering. With a market at hand for washed sand and gravel it is preferable to use such material rather than to ship better clay from a distance.

Edward J. Tournier, mechanical engineer, has admirably described and illustrated the mechanical equipment used in the large-scale sintering of metal-bearing products in *Iron Age* of Jan. 19, Feb. 16, and Mar. 8, 1928. Substantially the same equipment will be used in the manufacture of aggregates. They have been made in full size equipment and with the regular operation force, at a large metallurgical plant, and probably will reach the market in quantity within a year under the trade name of Lytag.

In the *Journal* of the American Concrete Institute (March, 1931), Frank A. Randall, structural engineer of Chicago, has discussed in considerable detail, in terms of money, quantities and weights, the economics of light-weight concretes in buildings. He discusses concretes weighing from 50 to 150 lb. per cubic foot, and their effect on the cost of the framework and foundation of buildings ranging in height from 5 to 30 stories.

M. Roberts, Seattle, Wash. (written discussion).—Mr. Hughes' paper is such a timely one that it is likely to lead to much inquiry into the subject. To those who already are interested in concrete of less than ordinary weight it provides an up-to-date survey of authoritative character.

The author's first division of the materials available for use as aggregates is headed, "Naturally Occurring Aggregates." Under this heading it appears that only one group is of present importance; namely, volcanic cinder, tuff and pumice. The author states that, "The deposits of such rocks are confined to the Rocky Mountain and Pacific Coast states." On the Pacific Coast there are so many deposits of these types with useful possibilities, and the rocks possess such varied characteristics, that it seems proper to call attention to them. Fortunately some of the occurrences are near lines of railway.

Material of the type classed by the author as volcanic cinder is found at many places in the Coast states. Extending from the Cascade Range eastward through southern Washington and northern Oregon, and continuing on into southern Idaho, is the Columbia River lava field, which covers thousands of square miles and ranks among the largest lava fields in the world. To the south of it, in central and southern Oregon, northern Nevada and northeastern California, are numerous other areas where products of vulcanism are prevalent.

Among the flow rocks of wide variety, the bedded deposits of ashy character, and the pumiceous material found throughout this portion of the Great Basin, are many occurrences that are interesting from an economic standpoint. In the Yakima Valley in south central Washington, quarries have been opened at a number of points near the main line of the Northern Pacific Railway, to meet local demands for stone that is light to handle and can be worked readily. For a preliminary investigation of the field, these quarries serve a useful purpose, since they afford fresh exposures of the deposits at some depth and also convey an idea of the strength of the stone and its behavior when worked. Diatomite also is mined in this region.

The lava fields in southern Idaho and eastern Oregon are traversed by lines of the Union Pacific system. The Oregon Trunk Railway of the Great Northern, extending southward from Celilo on the Columbia River through Bend in the center of Oregon to Klamath Falls, with a proposed extension into California, taps a region in which all three types of volcanic material are found and in which diatomite is quarried.

Besides the northeastern portion of California, other areas of interest in that state are found scattered along its easterly part, also in Lake County in the Coast Range, and in other regions. In many localities small quarries have been opened in these rocks of light weight. In Santa Barbara County diatomite is extensively quarried.

In the course of a survey made for a housing corporation in Boston, which has pioneered in the field of light-weight building units, I examined and sampled about one hundred deposits situated in the several regions just mentioned. As in the case of mineral prospects, numerous occurrences were examined for every one that proved interesting. When a deposit seemed to be unavailable it was usually so classed for one or more of the following reasons: (1) distance from transportation at the present time and probably for some years to come, (2) variation in the quality of the material, (3) uncertainty as to the quantity present of a particular grade, (4) weakness of the material, or (5) too high specific gravity. However, many deposits were seen that appear likely to prove useful.

While some of these rocks of volcanic origin are not extremely light in weight, their specific gravities are much less than the average of the gravels and crushed stone that ordinarily are used in making concrete. Beauty of coloring in shades of pink, red and brown is not uncommon and has led to the use of rocks of such colors for building purposes. In the agglomerates the included fragments of odd shapes and varied colors produce bizarre effects, which can be brought out by dressing the stone or sawing it.

The line of stately volcanic cones rising from the Cascade Range includes Mt. Baker and Mt. Rainier among its northerly outposts, Mt. Hood and several lesser peaks in its central portion in Oregon, while Mt. Shasta and Lassen Peak in California mark its southern end. Each of these outlets has produced not only the vast mass of the cone itself, consisting of lava and in many cases of much agglomerate as well, but within the shadows of several of these peaks I have found also more or less tuff and pumice.

The quantity of pumice to be seen varies greatly among the dozen or so of principal cones in the Cascade Range. At some of them the showers of pumice produced only thin layers of fragments, perhaps a few inches in depth, so far as natural exposures

disclose. Certain of the Cascade volcanoes, however, emitted pumiceous material in vast quantities. Near Glacier Peak, in the north central part of Washington, the surface has been coated with pumice over an area of several square miles. Owing to their lightness the fragments are easily moved by surface agencies, with the result that in places the pumice has accumulated on slopes and in gulches to depths of many feet. This region lies a few miles north of the transcontinental line of the Great Northern Railway.

A region in southern Oregon that extends from Crater Lake toward the east for a number of miles, and for shorter distances in other directions, presents one vast field of pumice, which dwarfs all others on the Coast. The topography is varied and includes both sharp, rocky ridges and many square miles containing flats and irregular surfaces with mild relief. Over this area is spread a thick coating of pumice fragments that range in size from dust to one foot or more in diameter. The heavy forest of conifers that is so characteristic of the Cascade Mountains appears here to be stunted by the lack of soil and the porosity of the filter-bed of pumice.

The Cascade line of the Southern Pacific Railroad between Eugene, Ore., and Black Butte, Calif., which was opened in 1927, passes through this field of pumice from north to south. Between Lonroth, at an elevation of 4666 ft., and Kirk at 4533 ft., 46 miles farther south, railway cuts that are wholly in pumice show it to be at least several yards in depth, while wells ranging from 25 to 50 ft. deep, and occasionally from 75 to 100 ft., have failed to pass through the deposit. Certainly its depth is great enough to make it readily workable over large areas.

The average size of the pumice particles varies in different parts of the field, depending on the distance from the source and on other factors, but in general the size diminishes toward the north and east. This variation could be used to advantage in choosing a site for production. At a given place the sizes vary considerably, yet a great majority of the particles at that place would fall within moderate limits of size. A screening operation could be devised to remove the fines and the occasional large lumps, and deliver a product or products having considerable uniformity of size.

A concrete composed of cement with an aggregate of coarse pumice and a sand of pumice obtained by screening or by grinding small fragments would have such extremely light weight that it might meet some special need. Diatomite also affords possibilities of this sort. In the mining of diatomite for such exacting uses as the filtration of oil and of sugar, varying amounts of lower grade material are produced which in part form a waste product. Some of the impure material possesses more crushing strength than the pure diatomite. If use can be made of this product, the quarries in Washington, Oregon and California offer an immediate source of production.

R. W. Smith, Atlanta, Ga. (written discussion).—I should like to call attention to the shale deposits of northwest Georgia, suitable and well located for the manufacture of Haydite and Lytag for the large market of the southeastern states. These deposits are found in practically every county in northwest Georgia from Cartersville and Rome north to Chattanooga. They are within easy shipping distance of Atlanta, Birmingham, Chattanooga, Macon, and other cities which will probably have a large industrial growth in the future. These shales will be described in a report of which I am the writer, to be published soon as *Bulletin* 45 of the Georgia Geological Survey.

E. Christensen, New York, N. Y. (written discussion).—The paper by Mr. Hughes is an admirable survey of the light-weight aggregate industry, and such a survey is timely. During the past 10 years or so there has been a steady growth of the understanding of our need for aggregates, natural or artificial, that would produce lighter concrete. And it has been discovered that such aggregate, because of its cellular structure, contributes other and most desirable characteristics to the concrete. Concrete has ceased to be an inert mass, having strength and density as its sole attri-

butes; instead we think of a concrete having insulative properties, absorption of sound, etc., and we have taken a new view of the problems of weather resistance and moisture movement.

The American Concrete Institute, at its convention next week, will devote a whole session to the problems of light-weight aggregates, and at that session I shall have the pleasure of presenting a rather lengthy paper[1] on cinder concrete—this being the matter which has occupied me for 10 years or so.

The paper by Mr. Hughes shows very clearly that light-weight aggregates are with us today, and that they are here to stay. It is possible that *the* light-weight aggregate has not yet been developed, but the line of development has been determined.

I believe that I am justified in saying that the cinder concrete industry has taken the lead. How long we shall keep it is hard to tell. Cinders form a cheap and satisfactory aggregate, and cinder concrete within its proper field of application is no longer to be considered a "substitute concrete" but a light-weight concrete of definite and desirable character. It is making the tall buildings of New York possible—one of these buildings alone may well have 25,000 cu. yd. or more in its floors. In the form of masonry units it is putting concrete on a competitive basis with such materials as clay tile. We are well above the 30 million large cinder units per year—and the field of application is steadily expanding.

The cinder concrete industry wants the right kind of cinders—not the household ashes that some of us may be thinking of now. When writing the paper mentioned, one of my first problems was to present a definition of "cinders." It is: "The residue, containing more or less carbonaceous matter, from high-temperature combustion of coal or coke—known as 'industrial cinders,' 'boiler cinders' or 'steam cinders,' to the exclusion of the residue from domestic furnaces. Forced draft is generally applied in the combustion process."

The consumers of large amounts of coal should be interested in producing that kind of cinders, and in keeping the cinders free from contamination. The quality of the aggregate is mainly determined by the nature of the fuel and the method of burning. To obtain the same quality of concrete, one cinder may well require 50 per cent more cement than another cinder—and the maker of concrete cannot afford to waste his cement. A good quality of cinder is not a refuse—it is a commodity that the maker of cinder concrete wants. The producer of cinders should give some thought to that.

As pointed out by Mr. Hughes, the cinder unit industry has grown rapidly under the leadership of the so-called "Straub patent." Veteran engineers may question the validity of such a patent at sight. The courts have broadly sustained it. Regardless of the merits of the patent, it has been instrumental in keeping the industry under a certain control, and it is imperative for the common welfare to prevent the ignorant and unscrupulous maker of concrete from using cinders without discrimination. There is a "trick" to making good cinder concrete—it cannot very well be done in a back yard.

As I said before, the first word has been said, but not the last. There is a great field for further development.

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[1] E. Christensen: Cinders as Concrete Aggregate. *Proc.* Amer. Concrete Inst. (1931) **27**, 583.

The Chivor-Somondoco Emerald Mines of Colombia

By P. W. Rainier, Minas de Chivor, Guateque, Colombia, South America

(New York Meeting, February, 1930)

The Chivor emerald field is situated on the eastern slope of the Andes in the Department of Boyacá, at an elevation of about 8000 ft. above sea level. It overlooks the Llanos (plains) of the Orinoco and is watered by many streams flowing into the Rio Meta, the Orinoco's largest tributary. The Rio Meta is navigable from the Orinoco to a point within about 50 miles of the Chivor district and may eventually prove to be the reasonable outlet for the future development of this part of the country.

The topography of the district is extremely rugged and mountainous (Fig. 1), the difference in elevation between two points on a mining claim 1 km. square being sometimes as much as 4000 ft. The ground is covered with dense forest growth and the available timber is suitable and sufficient for mining purposes, but it is becoming gradually depleted by Indian clearings as the country around the mines becomes settled. The many streams combined with the rough topography give good possibilities for hydroelectric power.

The center of population nearest to Chivor is Bogotá, the capital of the republic and a city of some 300,000 inhabitants; the distance is two and one-half days' travel. The first stage of the journey from Bogotá is accomplished by rail to Chocontá, some 4 hr. travel. From Chocontá the journey is made in the saddle, one night being spent in Guateque. The latter is a small town and the nearest post office to the mines, being some 10 hr. ride distant. The roads are mere bridle paths, in parts hazardous and almost impassable in the wet season. All supplies brought to the mines, other than those produced locally, are carried by pack mule; mule freight from Chocontá costs 3 centavos[1] per pound.

The climate of the Chivor district is healthful and bracing. The wet season begins in March and concludes in November, the remaining months being dry. The annual precipitation is about 150 in. The temperature throughout the year is nearly constant, varying from about 45° F. to about 65° F. shade temperature. The rains are cold and unpleasant, and for the greater part of each day in the wet season the country is shrouded in dense mist. A fire can with comfort be kept burning in a living room all the year round. There are a few slight

[1] At the present rate of exchange (October, 1929) 100 cents U. S. currency is equivalent to 98 centavos Colombia.

thunderstorms during the rains, particularly in August and September, but strong wind storms are practically unknown.

Early Mining Operations

The Chivor mines were worked by the Chibcha Indians for a long period before the Spanish Conquest in the sixteenth century; as they were the only source of emeralds known to the Indians at the time of the conquest, it follows that all the emeralds looted by the Conquistadores (an unknown but undoubtedly very large quantity) must have been mined here. The only other known deposit of emerald in Colombia of any importance is in the Muzo district, about 100 miles northwest from Chivor; the Muzo deposit is controlled by the Colombian Government and is not at present being operated.

After the conquest, Chivor was worked by the Spaniards, evidently on a considerable scale. Many of their old workings recently exposed show that their method of mining differed little from that in use today. Toward the end of the sixteenth century Chivor was abandoned by the Spaniards, because they had practically exterminated the local labor supply; the district became depopulated and overgrown with forest and the location of the mines was lost. They were rediscovered by Don Pacho Restrepo, a Colombian, about the beginning of the present century, after many years of search.

Production Gradually Increasing

The Chivor field was operated intermittently and changed hands frequently from the time of its rediscovery until 1925. At present it is controlled by Colombia Emerald Development Corpn., an American group, which has been operating successfully and on an increasingly large scale since it acquired control in that year. Prior to that date it was generally regarded as capable of only low-grade production and part-time operation—the latter by reason of an insufficient water supply in the dry season.

Table 1.—*Production of Emeralds at Chivor Mines*

Year	Number of Carats Mined					
	Color 1	Color 2	Color 3	Color 4	Color 5	Total
1926		3,170	400	11,500	28,400	43,470
1927		4,592	11,936	15,554	5,443	37,525
1928		505	10,668	4,299	7,240	22,712
1929 (first half)	200	4,985	10,135		120	15,440

The present operators have, in their 4 years of development, produced in increasing quantities emeralds of the higher grades and have proved up a body of formation containing high-grade emeralds that is sufficient for many years' operation. The ditch line now extends some 10 miles, tapping many streams hitherto unexploited and giving an ample water supply all the year round. The quantity of annual production is necessarily irregular in this kind of mining, governed as it is largely by the size of pockets encountered during the year's operations; however, Table 1 will illustrate the improving quality in the 4 years of operation.

Occurrence and Grades of Emeralds

The geology of the Chivor emerald occurrence was not scientifically investigated until 1926, when Charles Mentzel made a study of the geology and emerald occurrence; the geological information in this article is given with acknowledgment of his very complete report. Subsequent development has shown his findings to be correct, and they are, in great degree, responsible for the present successful operation.[2]

The emeralds occur in veins in a thick bed of shales. The veins are small, rarely attaining 3 in. in width, 200 ft. in length or 100 ft. in depth. The principal gangue minerals are pyrite and albite, the former having preceded and the latter followed the emerald mineralization. The emeralds are found in shoots or pockets in the gangue minerals; these shoots are rarely more than 3 ft. deep and may extend the length of the vein, being usually horizontal. Occasionally several shoots occur one above the other in the same vein. The emerald mineralization is very irregular and many barren patches occur in the veins between shoots or pockets. The emerald was evidently deposited from solution; where conditions were favorable it crystallized and formed the clear emerald, while elsewhere it appears as the green opaque emerald mineral, locally known as *morralla*, which is of small commercial value. The color of the clear emerald crystals varies from pale to dark green, depending on the amount of chromium oxide present; the value of the stone increases directly with the depth of color. The color of the emerald is usually consistent throughout a shoot and the amount found in a single shoot has varied from a few carats to 30,000 carats. A series of shoots in the same vein are almost invariably of the same color and in some parts of the property a series of parallel veins carry emerald of identical color; these form a zone or area in which the color and therefore the quality of the production to be mined is known.

The sales value of the emerald is determined by its color, brilliancy and freedom from flaws. The color is most important; for instance, a

[2] Appendix 1, page 213.

5-kt. stone of good brilliance and free from flaws would appraise for about five dollars if it were of Color 5 (very pale green); whereas the same stone, if of Color 1 (very dark green) would appraise for hundreds of dollars per carat. Stones have been mined varying in size from minute crystals up to several hundred carats in weight, and it is probable that even larger stones will be found with further development.

Mining Methods

The method of mining is extremely simple. The forest growth on an area to be exploited is first cut down and burned, this being done pre-

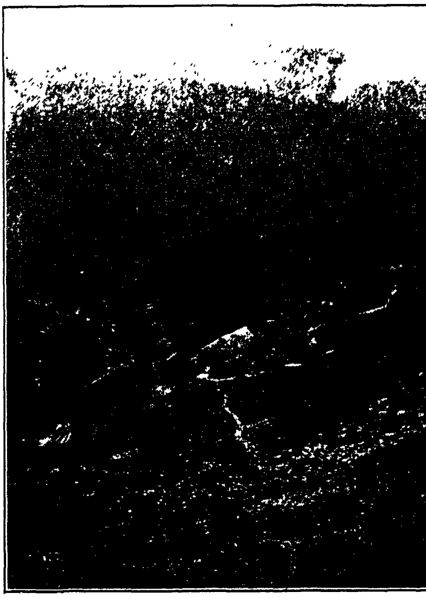

Fig. 1.—Part of labor camp at Chivor mines.

Fig. 2.—Cutting down steps.

ferably in the dry season. A trench is then dug in a direction to crosscut any veins that may lie below the surface, the loosened dirt being washed away by a head of water released from a tank above. These tanks, or *tambres* as they are called locally, are of simple construction; usually a *tambre* is a mere excavation with a wooden gate which is raised by a lever; the capacity is usually from 6000 to 10,000 cu. ft. The tambre gate is sufficiently large to allow the tank to be emptied in less than one minute, giving a big rush of water. As the trench deepens the sides are cut down in steps (Figs. 2 and 4). When the miners have cut a flight of steps to the bottom of the slope, they retrace their mining operations by knocking down the steps already formed. When an angle of about

FIG. 3.—TYPICAL V-SHAPED CUT.

FIG. 4.—CLOSE VIEW OF CUTTING DOWN STEPS. THE SLOPE IS MUCH STEEPER T IT APPEARS TO BE.

45° is attained by the sides of the trench the bottom is deepened and the operation of cutting down steps and knocking them down is repeated. Thus a V-shaped cut is obtained, eventually of considerable depth (Fig. 3). As the sides are steeper than the angle of repose, all dirt loosened by the operation falls to the bottom of the cut to be washed away by the tambre. From the workings of the Chivor mines there is an almost vertical fall of thousands of feet to the valley of the Sinai Creek, making ideal conditions for spoil disposal by this method. Occasional hard ledges of rock are hand-drilled and blasted. Apart from the drilling, the only tools used are crowbars and hoes.

As an instance of the extent to which these operations are conducted, a recent crosscut executed in this manner was 400 ft. long, 120 ft. deep, and involved the removal of approximately 200,000 cu. yd. of material. Of this material a fair proportion was rock in which about 4000 lin. ft. of hole was drilled and 1000 lb. of 90 per cent. gelignite was used. In drilling emerald formation great care is necessary in the placing of holes in relation to emerald veins to avoid breakage of the gem crystals. The cost of the operation described, including rock drilling and explosives, was slightly under 10 centavos per cubic yard.

The men work in gangs of from 15 to 40. Each gang is handled by an Indian foreman and close supervision is maintained by responsible members of the staff. When emerald formation is reached, distinguished by firm to hard rock with well-defined veins, a close watch is kept for a showing of morralla, or emerald mineral. This is immediately reported to the company official supervising the work and he closely watches the development of the vein. The emeralds are picked by hand from the vein, cleaned with acid, graded and prepared for shipment.

Under the Colombian Mining Law, it is illegal to transport rough or uncut emeralds in the country except under government seal; therefore, when sufficient production has accumulated at the mine to make a shipment, a government inspector is sent for to Bogotá. This official weighs and seals the production, which is then taken to Bogotá. The production must be delivered in the government offices with seals intact, the seals are here broken, the emeralds reweighed and appraised by the government appraiser and resealed. This accomplished, an export permit is obtained and shipment abroad is made under seal.

Thefts Infrequent

Provision against theft of emeralds at the mines is simple and quite effective, the loss being negligible. The necessity of transporting uncut emeralds under seal, as just described, is in itself a great deterrent to theft, making it very difficult to dispose of contraband stones. The staff of assistants at the mines are men of known integrity and one of

them is continually watching the men at work wherever emerald formation has been encountered. An additional safeguard is the fact that the men work in gangs and a theft by one man would undoubtedly be seen by others and probably immediately reported.

As soon as a show of morralla is reported, the official supervising the work closely watches the development of the vein as the hanging wall is removed and the emeralds are picked under his close supervision and handed to him. At the close of the day's operation, or before the departure of the official, the producing part of the vein is sealed with wet clay upon which he writes his signature. The vein is then further closed by covering it with a considerable quantity of rock and dirt from the bank above. When the rock and dirt are removed, the signature on the clay seal is examined to make sure that the vein has not been tampered with. Should another show of moralla be encountered while the official is occupied with a vein, he orders the men to work in a different place until he can supervise the development of the new vein. A bonus is paid to the man who first reports a show of moralla if the vein thus reported develops into a producer.

A check on theft is made by offering a substantial reward for any stones found in the streams or discharges below the mines, thus giving a thief an easy means of disposing of his stones. In the last two years only one small stone has been presented for sale in this manner.

Labor Supply Variable

With the exception of the general manager, the staff of assistants at Chivor are Colombians of the better class. They are loyal, energetic and intelligent and are not the least factor in the success of the operation.

The chief problem of the Chivor field is a sufficient labor supply to maintain operations on an adequate scale. In the immediate vicinity of the mine the country is somewhat sparsely settled, although this condition is gradually being remedied as men are attracted to begin cultivation in the neighborhood by the market provided for local produce at the mines.

The bulk of the labor employed is drawn from the Garagoa and Machetá valleys, some 6 hr. ride to the north. These are large and fertile valleys, densely populated by descendants of the Chibcha Indians, who, indeed, constitute the bulk of the population of the Department of Boyacá. The population of these valleys is ample to provide a labor supply for the future development of the district. The Chibcha, however, is an agriculturist and only works at the mines when his *finca*, or little farm, does not need his services; this makes the labor supply seasonal. During the period from August to February, inclusive, labor is most plentiful. During this period the force at the mines is occasionally

as great as 200 men. During the rest of the year, labor is scarce, as this is the planting season for the various crops grown in the district. However, year by year, more men are realizing the advantages of steady work for wages; in this respect the trading store maintained at Chivor is a factor, providing them, as it does, with many of the minor luxuries of life for which money is needed. During the four years of operation by the present company, labor supply has improved as follows:

Year	Shifts Worked	Year	Shifts Worked
1926	13,994	1928	23,003
1927	14,231	1929, Jan. 1 to Aug. 31, incl.	18,000

The wage scale paid at the mines varies from a maximum of 60 centavos paid to miners to 10 centavos daily paid to small hoe-boys. The average shift costs 40 centavos daily. In addition, the men receive free food and housing. The former is prepared for them by cooks paid by the company. The daily ration issued is as follows:

5 lb. roots (yuca, arecacha or plaintain),
1 lb. grain (maize, peas or beans),
1 lb. meat (usually beef),
1½ lb. miel (sugar cane molasses).

The beef is killed on the property after being inspected for disease. The miel is made into *guarapo* and issued as such, the daily ration being sufficient to make about 2 gal. of this drink. Guarapo is the national drink of the Chibcha and his capacity seems to be unlimited. It is not unlike a hard cider and quite palatable to civilized tastes. The alcoholic content is small and it is apparently healthful. All of the products named, except the beef, are grown locally and brought to the mines by pack mule. The cost of the complete daily ration is 40 centavos, making the total cost of the average shift 80 centavos. A 10-hr. day is worked.

Characteristics of Native Workers

The Chibchas are excellent laborers but, like most primitive peoples, need ruling with extreme firmness and, of course, with justice. Under these conditions they are easily managed and obedient; they are naturally a hard-working and industrious people. They are, on the average, intelligent but are naturally conservative and find some difficulty in learning methods which differ from their own. Better results are obtained by allowing them to do their work in their own way, whenever possible, than by forcing them to employ methods that are strange to them. Some make excellent underforemen but as a rule they are not sufficiently developed to be fitted for more responsible positions. Usually they are

completely illiterate and few are sufficiently educated to sign their names. On the whole, they are excellent unskilled workers, but the possibility of their being able to learn skilled trades is limited. Their speech is the Spanish of the old Conquistadores.

In temperament the Chibchas are normally cheerful and good natured and have considerable sense of humor; when aroused they are prone to settle their differences with the *penilla*, or belt knife, which is carried by every man. They will show great respect for authority, when authority deserves that respect, and serious labor troubles are almost unknown on the field. Their morals are, on the whole, good; they are fond of their families and domestic scandals are rare among them. They are not prolific, three children being about the average family.

In physique the Chibchas are short and sturdy, with a great chest development, the latter probably due to the rarified air of the altitude at which they live. They are hardy and work all day in the cold rains of the wet season with no other coverings than shirt and trousers and a woollen *ruana* over their shoulders. They are fleet of foot and will carry a 60-lb. load over 30 miles of mountainous country in a day. They are petty thieves and inveterate beggars, but do not practice banditry; journeys through their country can be undertaken with safety and organized crime is unknown among them. They were converted to the Catholic faith by the Conquistadores, but many of their ancient superstitions remain fixed with their newer religion. They were originally sun worshippers.

Future Possibilities

On account of the inaccessibility of the Chivor field the possibilities of using mechanical means of excavation are limited and the advantages doubtful. It is unlikely that the present low cost of excavation by hand labor could be approximated by any mechanical means.

The Chivor emerald field seems to have great possibilities. The emerald formation is of practically unlimited extent and most of it can be worked at a reasonable profit, providing operations are undertaken on a sufficiently large scale, while very rich pockets are occasionally found. The water supply is ample. With the steadily increasing labor supply and continued intelligent development on an increasing scale, the Chivor district should eventually rank as one of the great gem-producing areas of the world.

Earlier Papers on Emerald Mining

J. E. Pogue: Emerald Deposits of Muzo, Colombia. *Trans.*, A. I. M. E. (1917) 55, 910.

R. L. Codazzi: Minas de Esmeraldes. *Bol. de Minas y Petroleos* [Bogotá] (1929) 1, No. 2.

Appendix 1—Geology of Chivor No. 1 Mine

By Charles Mentzel,* New York, N. Y.

The formation at the Chivor mine consists of a series of conformable sediments perhaps several thousand feet in thickness, principally limey shales of light gray color, with some local lenses of carbonaceous matter. There is a top member of hard gray fossiliferous limestone and a lower member, rarely exposed, of hard blue thin-bedded limestone or lime shale. All these have a nearly uniform dip of about 35° to the west and strike about N. 30° E. It is reasonable to suppose that the tilting of the sediments was the result of the mountain building of late Cretaceous or early Tertiary time. The folding was uniform although locally soft beds were greatly distorted. Economically the latter are unimportant. The rocks were jointed by the pressure, and later these joints were affected by faulting. The effect of these forces has been to crack the rock into wedge-shaped forms, readily removable by crowbars.

Fissuring

In places, there are vertical or steeply inclined fissures, striking about east-west and with a steep dip, generally to the north. After this fissuring, the beds were subjected to normal step and block faulting, extending across the width of the property. The faults dip 35° to 40° to the east, and strike in the same way as the sediments. The throws vary from a few inches to 60 feet.

The fissures that were formed by the faults are the most important sources of the emeralds. After the main faulting followed by mineralization, there was minor movement along the same fault surfaces, which fractured crystals in weak parts of the veins.

Mineralization

Mineralization occurred both before and after the faulting, in the following order: quartz, pyrite, emerald and albite. The most striking evidences of mineralization from deep-seated solutions are large veins and masses or beds of limonite, occasionally hematite, at times with large cubes and pyritohedrons of the pyrite of which they are the alteration product. In the iron masses are large crystals of quartz up to 6 lb. in weight, and as the quartz never contains inclusions of pyrite, it is reasonable to assume that quartz was the first mineral deposited. This view is strengthened by the fact that always above the iron bands are found masses of silicified shale, the interior of which may be unaltered. The pyrite (afterwards altered) was introduced after the faulting, which

* Mining Engineer.

is shown by the fact that never do the iron bands show evidence of faulting. The emerald deposition followed the iron; inclusions of pyrite crystals in emerald are not infrequent. There were probably two main periods of emerald deposition; the first, in which the east-west fissures were mineralized, and the second after the north-south fault fissures were formed. This is apparent from the fact that while the east-west fissures usually carry quartz crystals, the north-south fissures never do. Albite was evidently the last mineral deposited; fractured emeralds are invariably cemented with albite, generally stained with limonite.

No testing apparatus was available at the field at the time of my investigation, but the following minerals were determined by sight:

Quartz.—Clean transparent prisms, occasionally with both pyramids, sometimes as large as 8 in. long and 4 in. diameter.

Pyrite.—Cubes and pyritohedrons from ½ to 3 in. dia. Sometimes massive and granular. Generally partly or wholly altered to limonite; in places, to hematite.

Fuchsite.—This green chrome mica was identified at one place above the iron band.

Calcite.—The usual transparent rhombohedral form is common in the cap rock above the emerald formation; only one small gray rhombohedron was found in the vein matter.

Apatite.—Possibly an occasional crystal in a vein. Identification uncertain.

Manganese.—Cross fissures frequently contain a black, earthy mineral with dark brown to black streak, which may be a manganese oxide. Never found in north-south fissures.

Fluorides.—Probable. Surfaces of emeralds are often etched, probably by fluorine liberated from some mineral by the action of sulfuric acid from decomposing pyrite.

Albite.—This mineral and pyrite are by far the most important gangue minerals. Generally granular or sandy, sometimes in tabular translucent crystals as much as ½ in. long. Altered in places to a pearly white silicate (allophane).

Emerald.—Found in all kinds of fissures, principally below the iron bands, but in greatest abundance in the important fault fissures. The crystals are hexagonal prisms, and occasionally one dome begins to form a pyramid, which may be nearly perfect, in small crystals. Usually such crystals are exceptionally brilliant. The color varies from pale to deep green, depending on the amount of chromium oxide present. The chrome evidently entered in spurts while the crystal was being formed, because the gems sometimes have bands of varying tints of green at right angles to the prism axis and sometimes the axial portion is white with a green shell of varying thickness forming the prism faces. The depth of color is not influenced by any other minerals in the vein or

on the wall rock, but it is, of course, influenced by the diameter of the crystal. In veins where the darkest stones are found the color is uniform. The color is entirely fortuitous, some veins producing dark stones and others light stones.

In places the emerald material is opaque green and either uncrystallized or only partly so. This is called *morralla.* Undoubtedly the gems crystallized from solution and under proper conditions the crystallization was complete, but when the solution froze morralla was formed. This belief is strengthened by the fact that partly formed clear crystals are found frozen to morralla.

Origin of the Emeralds

The emerald-bearing solutions came from a deep-seated source and the gems crystallized at moderate temperature and pressure in fissure veins. Wall rock never shows evidence of action of high temperatures and pressure. The fissures first were mineralized with quartz and pyrite, and in the spaces between those crystals the emeralds were deposited. Later a slight movement occurred along the fault planes, which at places fractured the emeralds. Albite, last introduced, entered the fractures and filled the open spaces of the vein. In recent geological time, the decomposition of the pyrite stained the albite and the rock adjacent to the veins.

Size of Veins.—The veins vary in length from a few feet to 200 ft., and in width from a crevice to 8 in. The gems are found in shoots or nested in the veins, which may vary in productivity from a few stones to several hundred. In general, the stronger the vein, the greater the productivity.

Carbon Rock.—Owing to the fact that the rocks of the Muzo mines are carbonaceous, it is believed by some geologists that carbon has influenced both the amount and color of emeralds deposited. I have not had the opportunity to study the Muzo deposits, but my observations at Chivor lead me to conclude that carbon has no bearing on emerald deposition. Lenses of carbonaceous shale are the rule rather than the exception in fine-grained sediments, and are a nuisance. When the veins traverse the carbon lenses, which may occur in different parts of the emerald zone, the pyrite is undecomposed on account of reducing action of the carbon, and careful manipulation is required to liberate the emeralds from this hard mass. In the veins where the normal decomposition of the pyrite has taken place, the emeralds are found in soft limonite, and are readily picked.

Emerald Zone.—The zone of deposition of emeralds is bounded at the bottom by hard blue shale, in which wide fissures have not been able to form, and at the top by the iron bands, which seem to have acted as a dam, preventing in general the further rising of the emerald

solutions. In places emeralds are found above the iron bands, or between several iron bands, but these are usually limited in number or of very pale color, or of the nature of morralla, indicating that the main deposition was lower down. For practical mining purposes the iron bands may be taken as the key formation, under which is the place to look for emeralds. The thickness of the good zone varies in the parts examined from 100 to 600 feet.

Appendix 2.—Brief Review of Emerald Mining in Colombia

By C. Kendrick MacFadden,* New York, N. Y.

When the first groups of Spanish adventurers reached the highlands of Colombia, they were amazed at the vast stores of emeralds which had been treasured by the Indian tribes inhabiting that area. The reports of the most famous Conquistador, Gonzalo Jiminez de Quesada, were said to be replete with descriptions of the wonderful emeralds in possession of the natives, and one of his first efforts after his arrival on the High Plateau in 1537, when he subjugated the principal chieftains, was to ascertain the sources from which the Indians had obtained their gems. Many of the records of these explorers still remain in the ancient archives at Bogotá.

It is recorded that among the early remittances sent to the Spanish Crown by the Conquistadores were four chests of emeralds, many of them of large size, one of the large crystals having been cut into the form of a small cup.

It is evident that the native Indian chieftains realized something of the value of the emerald. Its hardness, brilliancy and ability to take a high polish had fascinated the aborigines. It had been mined by the natives in certain areas of the country for centuries. Many of the emeralds had been the subject of barter with other Indian tribes, this trade probably extending to the north as far as Mexico and to Peru and Bolivia on the south.

Throughout the past 300 years, despite the most diligent search, no other emerald-bearing areas than those in Colombia have been found in the Andean range, nor in Central America or Mexico. Therefore the so-called "Peruvian" and "Mexican" emeralds were in all probability the product of the ancient Colombian mines.

Extent of Emerald-bearing Areas

Intensive exploration throughout the emerald-bearing areas in Colombia has disclosed the fact that they are probably only to be found in a narrow strip of territory which extends from the vicinity of Muzo south-

* Director, General Mining Co.; Chairman, Bogota Syndicate.

east for a distance of about 100 miles to the foothills on the eastern flank of the Andes. The principal commercial deposits are in two localities, one known as the Muzo District and the other as the Somondoco or Chivor District. The geological characteristics of the areas are more or less identical, although the productive gangue at Muzo seems to be more heavily impregnated with carbonaceous material than that in the Somondoco section.

In addition to the emeralds found in pockets or veins, there are some in the float materials along the stream beds which drain the emerald districts, but these finds are inconsequential. Because of the brittleness of the gem crystals, when they become detached from the producing

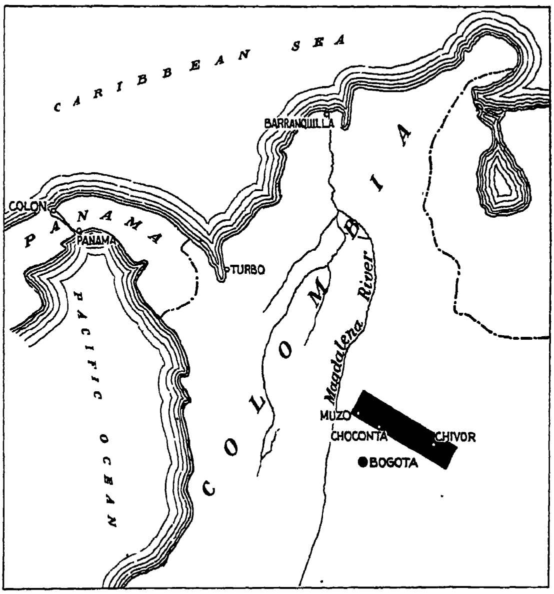

FIG. 5.—LOCATION OF EMERALD FIELDS IN COLOMBIA.

formations, through erosion or otherwise, they are soon ground into worthless chips by the gravels of the streams, which in the emerald-bearing areas are boisterous mountain torrents throughout the rainy season.

The emerald-producing areas so far discovered are covered with jungle growth containing heavy timber. At Muzo, as well as in the vicinity of the Chivor mine in the Somondoco section, the mantle of vegetation and forest has concealed surface outcrops to such an extent that the exploration and subsequent exploitation of much of the more desirable territory is of necessity a slow and laborious process. Some localities which have excellent possibilities cannot be worked economically due to the lack of adequate water supply. In this class of open-cut

mining large quantities of water are used to remove the debris which the miners dislodge. The necessity arises also for proper disposal of the vast amount of worthless material discarded and in several cases this particular problem has made commercial exploitation impracticable.

Geology and Chemical Composition

From early days geologists have attempted to correlate the structural conditions which are prevalent throughout the emerald-producing section but except along broad general lines no comprehensive geology has been completed.[3]

The chemical composition of the emerald as well as its form of crystallization is well known. The gem material is beryl deposited in hexagonal crystals ranging from a colorless transparency to the deepest emerald green depending on the amount of chromium contained. During crystallization many of the individual specimens become filled with feather-like flaws which are locally called *jardin* or "garden." Legend tells us that the ancient fortune tellers, by gazing deeply into the foliage of the "garden" enclosed in the emerald, could foretell with certainty happenings which were beyond the ken of the average person. Some of the emerald crystals, especially those obtained from the Somondoco district, have small bright crystals of iron pyrites included in the midst of the emerald material. These brilliant particles seem to be suspended within the emerald, which must have given the "crystal gazer" something of a puzzle to unravel.

Some crystals appear to be formed of layers of alternating light and dark green at right angles to the length of the specimens; others are formed with concentric bands of color—the almost colorless core surrounded by successively deeper green encircling bands—the outer layer being the darkest green. Again, crystals of good size and color will have penetrating and passing through them, at various angles to the main crystal, as many as three perfect hexagonal emerald crystals, often discernible only when viewed in a strong light.

Some of the crystals are twinned, some of triplet form, and often in a single "pocket" will be found an assortment varying in size and crystallization to a remarkable degree. A vein that contains at its outcrop a preponderance of stones of a given color or size will usually yield the same grade of emeralds throughout its length.

The emerald may be imitated by skillful artisans, usually with poor success. It has never been synthetically produced and the genuine stone may be immediately recognized by any competent mineralogist. The specific gravity of the Colombian emerald is from 2.7 to 2.8 and its dichroism is always very marked.

[3] See Appendix 1.

Uncertain Profits

The attitude of the Spanish Crown toward emerald mining is the same as that shown in connection with other classes of mines. In the beginning, the tribute to the Spanish Crown was approximately one-fifth of all mineral wealth obtained and this applied to the mining of emeralds as well as to the production of the precious metals. From time to time special grants or concessions for emerald mining were given to individuals favored by the government. Some of these grants, according to authenticated records, proved a bonanza to the owners but in the vast majority of instances the mining operations resulted in no real profit. It is known that in the early days when slave labor was procurable some operations yielded handsome returns, although even with slave labor the unusual hazards of this class of mining must have brought many disappointments.

One of the most interesting anecdotes regarding the early development described the operations of a titled Spaniard who had received, as his reward for services rendered the Crown, an exclusive mining concession for one of the famous deposits. He expended practically his entire fortune without result, although outcrops on the concession had indicated most attractive possibilities for the finding of rich pockets of emeralds. As his rights were about to terminate, and as a forlorn hope, he followed the suggestion of one of his laborers and tore down what had been supposed to be a worthless mass of formation which had shown no surface indication of emerald-producing veins. To his amazement he uncovered a spectacular deposit and within a few days extracted emeralds of value exceeding his entire investment. He selected a large parcel of emeralds from the lot and sailed to England, expecting to sell them at top prices to the lapidaries of London, Brussels and Amsterdam. He fitted up a room in his tavern in London and invited all his prospective customers to a magnificent banquet, after which he exhibited his collection of emeralds on a large table, amid the most appropriate setting, and confidently expected to obtain from them an offer for immediate purchase. The gem dealers were astonished at the quantity and quality of the assortment. They asked him, cautiously, whether there were any more such emeralds to be found at his mine. In an attempt to brag of his achievement he announced that all the emeralds had been obtained with a few days' labor and that many more gems remained unmined. The effect on his audience was immediate—they visualized the certainty of a large production of fine emeralds about to be thrown on the market and refused to make a bid. The seller attempted, in vain, to correct his falsification and it is said that his emeralds remained unsold for nearly two years, during which time the purchasers had satisfied themselves that the gems offered were likely to be the only product from that particular property.

Accidental Discoveries of Single Stones

The emerald, probably more than any other gem, intrigues the mind of the itinerant prospector. In its rough form it is an object of beauty; its identification as a valuable gem is immediate. It is often found in a soft or friable formation which, by weathering, exposes fine gem material. Even the most ignorant Indian of the highlands of Colombia is aware of the fortune that will come to him from the discovery of a fine emerald crystal. The history of the country is filled with stories of poor natives who thus unexpectedly find themselves in possession of an emerald of great value.

Throughout the emerald-producing areas, the craw or gizzard of every barnyard fowl that is killed for food is carefully examined, often with profitable results. As many as seventeen small emerald pieces have been recovered from an observant hen raised in the emerald area. The brilliant green particles were evidently selected by the fowl from the less attractive gravels on the range. At Muzo mine, during government operation, every fowl killed throughout the district had to be delivered by its owner to the police for examination of the viscera, under heavy penalty for noncompliance.

During recent years, the Colombian Government has attempted to nationalize all emerald deposits and the sale of rough emeralds has been prohibited, except from certain mining claims, titles to which were perfected many years ago and whose rough emeralds, on being presented to the proper government official and duly certified as to origin, are permitted to be sold; otherwise any rough emeralds discovered throughout the Republic are the absolute property of the government and subject to confiscation. In a case where it is proved that the emerald has been found on the surface, in areas which are not already under concession, the finder is given a proper compensation and it becomes the property of the government.

Present Production and Value of Emeralds

At present, there is but a single emerald deposit in Colombia being developed, this being the Chivor mine, supposed to be the one operated by the Indians before the conquest of the Spaniards. This property, situated in the Somondoco district, is owned by an American company and is the only mining venture of this class that is being systematically developed.

The other great mine from which large production has been obtained intermittently for the past 300 years is known as the Muzo mine. The greater portion of the Muzo deposit is controlled and owned by the government, as is also a neighboring deposit in the Muzo district known

as the Cosquez mine, previously operated at great profit. Although no accurate record can be obtained as to the total output of the Muzo district, it is known that the Government of Colombia in certain years has received more than one million dollars from its participation.

The emerald material is sorted by an expert immediately after mining, into five classes or grades and *morralla*, which is a semicrystallized product having much of the appearance of turquoise matrix, but green in color. This material at present is given no commercial value, but has possibilities for use in the manufacture of cuff link settings, and so forth.

No accurate estimate or prediction can be made of the proportions of the several classes in the total mined material. A fairly typical mining return sheet from the Muzo mine covering two months' operations (Table 2) shows percentages of the five principal classes into which the product was divided. The total weight of emeralds recovered was about 10 lb. Avd.

TABLE 2.—*Two Months Mining at Muzo Mine*

Classes	Weight, Carats	Value[b]	Per Cent. of Total Weight	Per Cent. of Value
No. 1[a]	523	$130,750	0.75	7.70
No. 2[a]	2,182	$218,500	3.15	12.80
No. 3	9,548	$477,400	13.60	28.60
No. 4	12,649	$316,200	18.50	18.50
No. 5	44,116	$551,400	64.00	32.40
	69,018	$1,694,250	100.00	100.00

Average per month, 34,509 cts. = $847,125 as operated by English Mining Co.

[a] It will be noted that the No. 1 and 2 grades, having a total weight of less than 4 per cent. of the output, yield more than 20 per cent. of total value.

[b] Estimated values based on one carat (3⅙ grains): No. 1 grade, $250; No. 2, $100; No. 3, $50; No. 4, $25; No. 5, $12.50.

The values given are believed to be considerably higher than are at present used by the government appraisers in estimating the value of the rough material.

It will be noted immediately that notwithstanding the very high value placed on the No. 1 grade, the total value of this grade mined during these particular two months represents less than 8 per cent. of the total values, whereas the No. 5 quality, valued at one-twentieth the price per carat, yields nearly one-third of the total value of the two months' mining.

The United States provides the best market for the superfine grades, although those classed as No. 2 and even No. 3 find a ready sale at satisfactory prices. The material of cheaper quality has always found a better market in the European countries and this is especially true of India, where the lighter colored stones seem to be in steady demand. In New York the superfine Colombian emeralds are sold at retail as high

as $3000 per carat and for some special gems as much as double this amount, thus far outranking the diamond or ruby in value.

It is interesting to note that even during a period when veins containing emeralds of fine color are regularly producing, an almost infinitesimal portion of the total product would grade as superfine gems, the proportion being something like 0.01 per cent. The exact percentages can only be ascertained after the stones have been cut by a skillful lapidary.

The mining manager of the British company which some years ago operated the Muzo mine on special contract with the Colombian Government is authority for the statement that once, after a long period of poor returns, one of the veins "commenced to produce;" a pocket or "nest" of high-grade crystals was encountered and from a few cubic yards of vein material, $400,000 in fine emeralds was obtained in a few hours. Such instances, although rare, are the incentives which urge the mining engineer familiar with gem mining to unravel the geological snarl which at present marks the genesis and deposition of the Colombian emeralds.

The lapidaries who make a specialty of the emerald, or any other colored gem, have as their ideal achievement the production of a cut gem exhibiting the maximum of color combined with as much brilliancy as possible. This calls for skill of a different order than that used in the cutting and polishing of a diamond, whose maximum value depends upon brilliancy alone, provided the "rough" is of superfine quality. The lapidaries who cut and polished the emeralds in earlier days did not possess the skill of the modern gem cutter, who, if he be an artist, may often greatly increase the value of an old emerald by recutting. The weight of the finished gem will be less than the original, but the increased value per carat may add as much as 100 per cent. to the modern appraisal because the color and brilliancy have been enhanced.

In Colombia one may be offered emeralds that have been handed down from generation to generation, which to the inexperienced seem to offer attractive speculative possibilities for purchase. As a general rule these emeralds of ancient cut lack the qualities desired today. It is evident that grading of emeralds is more of an exact science today than 100 years ago. Properly graded emeralds can be purchased more cheaply in New York, London or Paris than in Colombia.

DISCUSSION

(Donald M. Liddell presiding)

D. M. Liddell, New York, N. Y.—The discovery of the Chivor-Somondoco mines seems to indicate that one cannot dismiss stories about ancient mines. It pays to look around.

C. K. MacFadden, New York, N. Y.—That is an interesting comment. I know of nothing that intrigues the average American more than to hear the stories of some of the remarkable circumstances surrounding the discovery and development

of the early mines of some of the Latin American countries. I remember when this story of the ancient mines of Somondoco was told me by Christopher E. Dixon, the former manager of the Muzo property. I had been inquiring whether there was any other locality in Colombia where emeralds were to be found. He gave me the story of the mine and how it had been abandoned. It struck me at once as being a good mining prospect; some of my friends joined me and we purchased the several claims which included the old workings. Fortunately it has turned out to be a mine of great promise.

I have seventeen small emeralds that were found in the craw of a chicken that had been raised at these mines. At Muzo, the government mine, anyone who wishes to kill a barnyard fowl must take the bird to the police department; there they kill it and remove the gizzard and the craw for future examination. There have been fowls, I am told, that have had several hundred dollars worth of emeralds in their craws, undoubtedly collected in a few months from the surface sands and gravels of the district.

A. T. Wilson, London, England. (written discussion).—My technical knowledge is not such as to enable me usefully to comment on the paper, but I do know about the admirable organization developed at Chivor by Mr. Rainier. I have never seen a camp better run: food supplies for laborers, medical arrangements, hutments, all well thought out, effective and economical; and an excellent spirit prevailing throughout.

Some Relations of Ore Deposits to Folded Rocks

By W. H. Newhouse,* Cambridge, Mass.

(New York Meeting, February, 1931)

During the past few years the writer has been impressed by the close relation of many epigenetic orebodies with anticlinal structures. In the literature on ore deposits there is occasional mention that an oreshoot or district is on an anticline, but in most of the descriptions no weight or significance is attached to the relation, and usually the fact is concealed in a mass of structural detail.

A few writers have stressed the relationship in a few districts or on a certain type of deposit. Church, Spurr, Butler, Stahl and Schuette have called attention to certain features, but the general extent of such a correlation has not been made known in any publication that the author has seen. A recent article[1] on the localization of ore deposits, by well-known geologists, does not consider this common association.

The author has seen in the field a number of the structural features described, and in addition has made a fairly complete review of the literature. This paper presents enough of the examples found to show the frequency of the relation to anticlines, and since only a few deposits on synclines are known, all of these of any importance of which the writer knows are mentioned.

With certain qualifications which are discussed later, epigenetic ore deposits of replacement or vein type are mostly found where local upwarps of the earth's crust have taken place.

Where formations antedating the ore deposits are present in the proper attitude to demonstrate vertical warping (*i. e.*, more or less horizontal beds) the structural feature on which a mineral district, or mineralized area is located is almost always anticlinal, or domelike in nature. Some or even most of the ore deposits in a given district may be on the flanks of the upwarp, but few major districts are located definitely on downwarped areas. Synclinal basins within certain limits of size appear to be unfavorable to the type of fracturing that admits ore-bearing solutions.

Replacement orebodies frequently are localized in small anticlines or domes, the size of the anticline or roll often being more or less commensurate with that of the enclosed oreshoot. This feature certainly ranks

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[1] A. Locke and P. Billingsley: Trend of Ore Hunting in the United States. *Eng. & Min. Jnl.* (1930) **130**, 565–566, 609–612.

with faults, favorable horizons and impermeable roofs as one of the important elements of structure which may localize replacement orebodies.

Relation of Mineralized Districts to Upwarps

Several writers have pointed out the close relationship between large anticlines, or domes, igneous intrusion and ore deposits for certain mineral districts. Spurr[2] has discussed a number of districts in connection with his theory of igneous intrusion. Butler[3] points out the close areal relation of these features and their genetic significance in the ore deposits of Utah.

An examination of the literature on several hundred ore deposits has convinced the writer that the replacement and vein types of ore deposits are largely connected with upwarps, either with or without igneous rocks. Some districts, notably small ones, are not so located, and on many others structural evidence bearing on this feature is lacking. The following examples are illustrative of the scale and type of uplift or upwarp which has just been mentioned. The list could be extended almost indefinitely. These are selected to show the wide variations in types of ore districts included.

The frequent relationship of ore deposits to areas of uplift, or upwarp, is expressed in various degrees. In some, as in the Clifton-Morenci district, Arizona,[4] the relationship might perhaps not be regarded as a very close one. The ore deposits of Ray, Miami and Globe,[5] Arizona, when looked at broadly, are on or near the borders of a pre-Cambrian area bordered by Paleozoic and later rocks. In the areas mentioned the phenomena of intrusion and faulting are so obviously and closely related to the formation of the primary mineralization, and the structures are so complex, particularly in the Ray and Miami region, that one would hesitate to say that one of the prerequisites for the formation of the mineral districts was uplifting or upwarp. If there is such a relation in areas of this type, it is probably due largely to the well-known fact that stocklike igneous intrusion frequently is associated with upwarped or uplifted areas. A closer relationship is exhibited by the ore deposits connected with many other upwarps, and this seems to be particularly true as the size of the upwarp becomes smaller.

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[2] J. E. Spurr: Ore Magmas, **1**, 187–252, New York, 1923. McGraw-Hill Book Co.

[3] B. S. Butler and Others: The Ore Deposits of Utah. U. S. Geol. Survey *Prof. Paper* 111 (1920), 102–104; and under different districts, map opposite page 100.

[4] W. Lindgren: The Copper Deposits of the Clifton-Morenci District, Arizona. U. S. Geol. Survey *Prof. Paper* 43 (1905).

[5] F. L. Ransome: The Copper Deposits of Ray and Miami, Arizona. U. S. Geol. Survey *Prof. Paper* 115 (1919).

Other important districts located on anticlines, or domes, or on their flanks, are Park City, Cottonwood, Mercur and Tintic, Utah,[6] with the deposits near Eureka on the steeply dipping flanks of a fold, except the Iron Blossom ore zone which is in the syncline, and those of East Tintic connected with a dome, Bingham Canyon, with the oreshoots in sedimentary rocks on the flanks of an anticline, or according to some interpretations on an overturned syncline.

Coeur d'Alene, Idaho,[7] on the eastern flank of a large anticline, and the deposits of the Cranbrook area[8] in British Columbia, which includes the Sullivan mine, are divided into two areal groups and each group is located on an anticline. In the Salmon River district,[9] British Columbia, which includes the Premier mine, the mineralization is on the anticline of Big Missouri Ridge.

The Hedley mining district[10] is on the western flank of a large anticline, well up toward the crest where the dips are low, as compared with the dips in the western part of the area.

The Bawdwin mines, Burma,[11] are on the crest of an anticline which has been strongly sheared.

Other districts, which may be mentioned as showing similar features, are Ducktown, Tenn.; the San Juan dome, with four subsidiary domal or anticlinal structures on which are found four of the main producing areas with structural relations unknown to the writer in the fifth area; Goldfield, Nev.; Tonopah, Nev.; the southeastern Missouri lead deposits on the flank of a large dome; less important lead deposits in central Missouri also on a large dome; the Illinois-Kentucky fluorite deposits on a dome; the Shropshire, England,[12] lead-zinc deposits in two areas, on two anticlines; also the most important lead-producing area in Great Britain on the Pennine anticline, including the region of Derbyshire, West Yorkshire and an area to the north; and the lead-zinc veins south of Essen, Germany, in the Velbert anticline. To these may be added the ore deposits of Buchans, Newfoundland, and those of Broken Hill, Australia, and Sardinia, Italy.

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[6] B. S. Butler and Others: *Op. cit.* Cites much previous literature.

[7] J. B. Umpleby and E. L. Jones: Geology and Ore Deposits of Shoshone County, Idaho. U. S. Geol. Survey *Bull.* 732 (1923) 17.

[8] S. J. Schofield: Geology of the Cranbrook Map-Area, British Columbia. Canada Geol. Survey *Mem.* 76 (1915).

[9] S. J. Schofield and G. Hanson: Geology and Ore Deposits of Salmon River District, British Columbia. Canada Geol. Survey *Mem.* 132 (1922).

[10] C. Camsell: Geology and Ore Deposits of the Hedley Mining District, British Columbia. Canada Geol. Survey *Mem.* 2 (1910).

[11] M. H. Loveman: Geology of the Bawdwin Mines, Burma. *Trans.* A. I. M. E. (1916) 56, 170–194.

[12] R. H. Rastall: The Geology of the Metalliferous Deposits, 280–301. Cambridge, 1923. University Press.

Santa Eulalia, Mexico,[13] is on an arch, as are Laurium, Greece[14] (Fig. 1); Aspen, Colo.;[15] Matehuala, Mexico, and Velardena, Mexico.

Baker[16] says that "All the workable ore deposits noted by the writer in the limestone rocks of northeastern Mexico . . . are in anticlinal structures . . . generally along the axes of major anticlinal structures and subordinately in zones of minor crumplings within larger anticlinal structures."

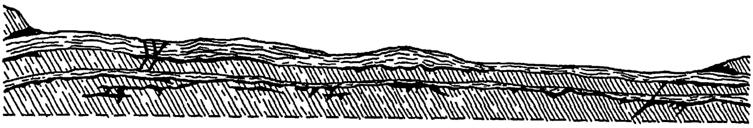

Fig. 1.—Part of main anticline, Laurium, Greece.
Oreshoots black, mainly under shale beds in limestone or marble.

Fletcher,[17] in writing on the silver-lead manto deposits in Mexico, says, "Most of these lead-silver districts are associated with anticlinal structure."

An interesting group of which the members in general do not show close igneous affiliations and are found in such structures includes the carnotite deposits of southwestern Colorado;[18] Silver Reef, Utah;[19] the Red Beds type of copper deposit in white Canyon, Utah,[20] which also contains uranium and cobalt, and Coro Coro, Bolivia, [21] on a broken anticline.

Of the copper deposits in the Permian of Texas, three are described on anticlines,[22] and in connection with this group may be mentioned the localization of several shoots of ore in the Lake Superior copper district[23] on the Allouez, Baltic, Winona and Mass anticlines, although it is said: "in detail the distribution of rich and poor ground is far more dependent

[13] B. Prescott: The Main Mineral Zone of the Santa Eulalia District, Chihuahua. *Trans.* A. I. M. E. (1916) **51**, 57–99.

[14] F. Beyschlag, P. Krusch und J. H. L. Vogt: Die Lagerstätten der Nutzbaren Mineralien und Gesteine nach Form, Inhalt und Entstehung, **2**, 285–287. Stuttgart, 1913. F. Enke.

[15] J. E. Spurr: *Op. cit.*

[16] C. L. Baker: General Geology of Catorce Mining District. *Trans.* A. I. M. E. (1921) **66**, 48.

[17] A. R. Fletcher: Mexico's Lead-silver Manto Deposits and Their Origin. *Eng. & Min. Jnl.* (1929) **127**, 512.

[18] R. C. Coffin: Radium, Uranium and Vanadium Deposits of Southwestern Colorado. Colorado Geol. Survey *Bull.* 16 (1921) and maps.

[19] B. S. Butler and Others: *Op. cit.*, 582–594.

[20] B. S. Butler and Others: *Op. cit.*, 619–622.

[21] L. de Launay: Traité de Métallogénie Gites Minéraux et Métallifères, **2**, 757. 1913.

[22] E. J. Schuntz: Copper Ores in the Permian of Texas. *Trans.* A. I. M. E. (1896) **26**, 97–108.

[23] B. S. Butler and W. S. Burbank: The Copper Deposits of Michigan. U. S. Geol. Survey *Prof. Paper* 144 (1929) 117 and maps.

on character of rock than on structural position." The Isle Royal shoot in this district is near the trough of a syncline. To this list may be added Kennecott, Alaska.[24]

Relations of Epigenetic Orebodies or Oreshoots to Impervious Basements, Impervious Roofs, Anticlines and Terraces

In order to throw some light on anticlinal relations, it seems advisable to discuss briefly a few structures of similar nature, which aid in gathering descending solutions and localize orebodies. These may be included under the terms impermeable basements, basins or troughs, and synclines. Following this will be a similar brief summary of the effect of impermeable roofs in localizing hypogene orebodies. The description of hypogene orebodies localized by small anticlines or domes will then be given.

Impervious Basements and Synclines—Effect on Descending Solutions

By inspection of the numerous diagrams in the literature which show orebodies formed by descending solutions, one finds that there is, as might be expected, a variety of structural conditions connected with the ore.

Troughs with impermeable walls[25] may be formed in several ways, among which is the intersection of dikes, with shale, slate or quartzite, to form a concave surface upward. Faulting also may produce a trough with both walls of the same or of different impermeable rocks; fault gouge may serve as one wall; and folding may produce synclinal troughs or basins.

Concentration on a terrace formed of impervious rock is a related type, while concentration above an inclined impervious basement probably is the result of solution gathering into a smaller channelway or conduit, with deposition, replacement or solution in larger amount at this place, probably because a greater number of reacting ions is present in a given volume of host rock.

The iron ores of the Lake Superior region afford excellent illustrations of concentration by troughs and synclines. Many illustrations of these have been given in the literature. Other residual iron and some manganese deposits show the same general features, although, where for example, they are found on an eroded limestone surface they appear at times as strongly developed on the pinnacles as in the basins of the corroded limestone. This may probably be explained by the assumption that the ore minerals remain inert under some conditions and migrate in solution somewhat in others.

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[24] A. M. Bateman and D. H. McLaughlin: Geology of the Ore Deposits of Kennecott, Alaska. *Econ. Geol.* (1920) **15**, 1–80.

[25] Impermeable is considered not as an absolute but only as a relative term.

Oxidized zinc deposits are found usually below the sulfide mass from which they were derived. They may form on terraces[26] or on an inclined impervious basement or show no such structural control and be located merely in limestone.

The zinc sulfate solutions appear to have a limited tendency to migrate in limestone, and such structural control as basin-shaped structures would afford is not as marked apparently as with certain other types of ores.

Thus far, in order to reduce the structures to the simplest terms for purposes of discussion, faulting and fracturing have been little mentioned. Faults may and frequently do localize the minerals formed by supergene solutions, to the complete exclusion of the relations just mentioned. An example of such modified control is in the iron ores of the Mesabi Range, Minnesota, which are found where the formations are warped into anticlines and synclines. The warping caused fracturing, which allowed easy access and circulation of the ground waters that formed the orebodies.[27]

Impervious Roof—Effect on Ascending Solutions

Numerous references may be found to lead and zinc sulfide ores in limestone with a roof of shale. Bedded deposits, or veins with spreading orebodies underlying shales, slates, sills or flat dikes are often regarded by geologists as being due to damming of the solutions by an impermeable roof. Cinnabar ores,[28] according to Schuette, very commonly show such relations. The relation has been described in textbooks and is well recognized, so no attempt will be made at a full description, although several varieties will be mentioned with examples of each.

A type familiar to students is that illustrated by the occurrence at Laurium, Greece,[29] where ore in part forms tabular masses underneath the shale beds (Fig. 1); also in the Palomas, Chloride Flat, Georgetown, Hillsboro, Lake Valley, Cooks Peak and Magdalena districts in New Mexico.[30] Numerous similar examples are well known and need not be discussed.

A variation within this group is found in the Magdalena district, where oreshoots are found in limestone. "In their maximum occurrence

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[26] G. F. Loughlin: Geology and Ore Deposits of the Leadville Mining District, Colorado. U. S. Geol. Survey *Prof. Paper* 148 (1927) Plate 65.

[27] J. E. Wolff: Recent Geologic Developments on the Mesabi Iron Range, Minnesota. *Trans.* A. I. M. E. (1916) **56**, 156–158.

[28] C. N. Schuette: Occurrence of Quicksilver Orebodies. See page 413.

[29] F. Beyschlag, P. Krusch und J. H. L. Vogt: *Op. cit.*

[30] W. Lindgren, L. C. Graton and C. H. Gordon: The Ore Deposits of New Mexico. U. S. Geol. Survey *Prof. Paper* 68 (1910).

in lenses below impervious strata along the crest of low arches"[31] the axis of the arches pitch down the main dip of the rocks. Other examples

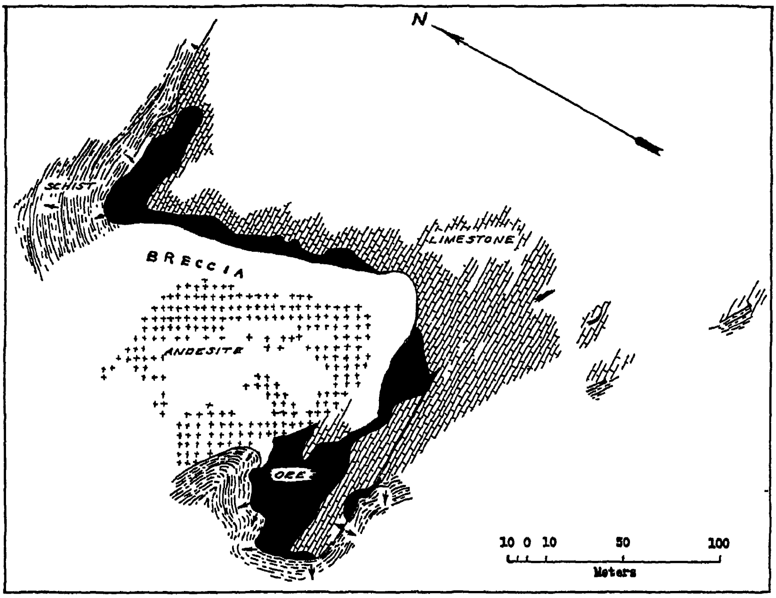

FIG. 2.—HORIZONTAL PLAN OF LEAD-ZINC OREBODY, STANTRG, YUGOSLAVIA.
Anticline pitches northwest, intruded by andesite. Andesite bordered in part by breccia, ore in subsidiary crests.

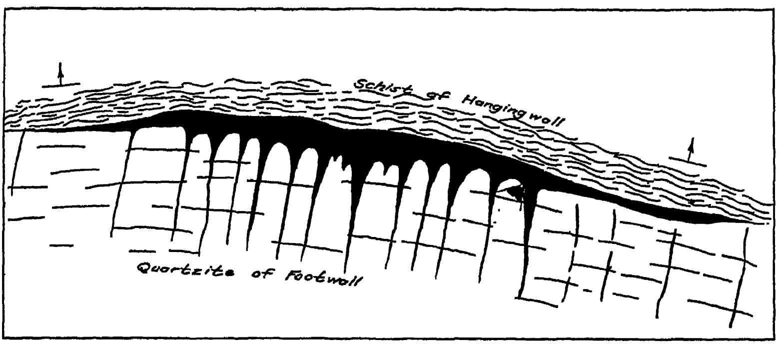

FIG. 3.—FERRIS-HAGGARTY MINE, ENCAMPMENT DISTRICT, WYO.
Ore in competent stratum on outer part of bend next to incompetent bed where tension has produced fractures. This type is probably common in limestone beds under shale, but with fractures masked by replacement.

affording illustrations of orebodies under impermeable rocks in pitching anticlines or arches are oreshoots at the Elkhorn mine, Elkhorn, Mont.,[32]

[31] W. Lindgren et al.: *Op. cit.*, 254.

[32] W. H. Weed: Geology and Ore deposits of the Elkhorn Mining District, Jefferson Co., Montana. *Ann. Rept.* 22, U. S. Geol. Survey (1900–1901) Pt. 2, 477–495.

in limestone under shale or slate, ore at the Stantrg lead-zinc mine, Yugoslavia[33] (Fig. 2) and the orebody at the Ferris-Haggarty mine, Encampment district, Wyoming[34] (Fig. 3), the copper ore being in brecciated quartzite in a steeply pitching arch underneath schist.

The United Verde orebody at Jerome, Ariz.,[35] was localized by diorite which "formed a steeply pitching inverted trough of relatively impervious material, such as would tend to draw together and localize the deep-seated solutions in their upward course."

Another type of variation is shown by the relations of an orebody at Rico, Colo.,[36] where a nearly vertical fissure passes up through a horizontal series of sedimentary rocks, branches or forms breaks across a wide zone on entering shales and the orebody is found localized under an impervious blanket of black shale, but remaining fairly closely within the broken or fractured area.

In summarizing the relations, we may conclude that there are three general types of relations of orebodies to impermeable covers: (1) a group related to local brecciation as well as an impermeable cover; (2) a type underlying impervious cover without apparent fracturing or brecciation of the replaced rock; (3) the inclined arched type of impermeable cover.

The first two types suggest that impounding of solutions has in some way facilitated replacement of the rocks, the last type suggests that the walls and material within a conduit carrying hydrothermal solutions have been replaced along a main trunk line of solution movement. The solutions appear to have a strong tendency to move upward in preference to a lateral direction, since the ore is formed in the high parts of the structure.

Anticlines, Noses and Terraces—Effect on Ascending Solutions

In the relation of ore districts to fairly large upwarps, and a similar relation of single oreshoots to small individual anticlines or domes, there appear to be all degrees of gradation in size. This will appear in comparing the following descriptions with those we have just considered.

The southeastern Missouri lead deposits are on the flank of a large domal structure, as has been pointed out by Spurr.[37] Also, Emmons writes: "A very considerable number of the largest deposits of lead ore are

[33] A. Brammall: The Stantrg Lead-zinc Mine, Yugoslavia. *Min. Magazine* (1930) **42**, 9–15.

[34] A. Spencer: The Copper Deposits of the Encampment District, Wyoming. U. S. Geol. Survey *Prof. Paper* 25 (1904) 72–82.

[35] L. E. Reber: Geology and Ore Deposits of Jerome District. *Trans.* A. I. M. E. (1921) **66**, 23.

[36] F. L. Ransome: Ore Deposits of Rico Mountains, Colorado. *Ann. Rept.* 22, U. S. Geol. Survey (1900–1901) Pt. 2, 291–293.

[37] J. E. Spurr: The Southeast Missouri Ore-Magmatic District. *Eng. & Min. Jnl.* (1926) **122**, 968–975.

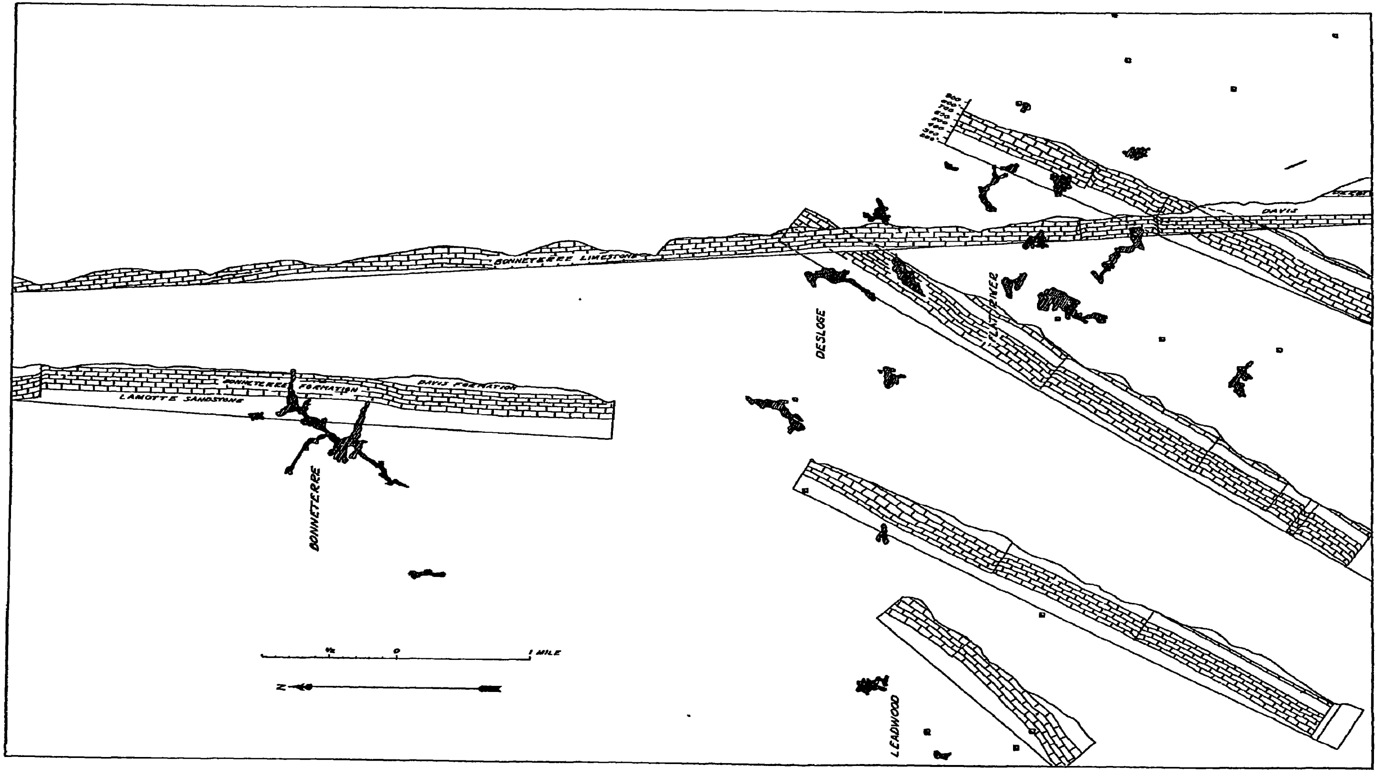

FIG. 4.—SOUTHEASTERN MISSOURI LEAD DISTRICT.

Plan of stoped ground (year 1908) and cross-sections of rocks. Base of section is line on plan through which it was taken. Same relative scales on sections as in original by Buckley, excepting Leadwood section. Ore deposits on anticlines.

found below gentle anticlines and near unconformities where the Paleozoic rocks lap against and dip away from the pre-Cambrian basement."[38]

The author wishes also to point out that the important deposits of Flat River and vicinity are on one minor anticlinal structure located on the larger structure, and the deposits in the vicinity of Bonne Terre on a similar minor anticlinal structure some five miles to the north.[39]

The ground between has not been important in ore production. Individual oreshoots seen by the writer in the Bonne Terre part of the district were mostly in smaller flat anticlinal arches. There is a relation then to folds of three or four different orders, one superimposed on the other. In plan, individual small projections of ore from a larger body were found usually localized in a small individual arch or dome, which in some cases showed no signs of fracturing in the well exposed roof.[40] It is apparent that the solutions have migrated to some extent, at least into some of the minor structures along beds; the arches retained ore in some manner, the whole being analogous in effect to a localization of petroleum by similar structures. In certain of these individual examples it seems certain that the ore is localized in shoots by the anticlinal structure as such, instead of being localized along fractures or in fractured rocks which had been caused by small folds. The folding is slight and dips are low. It is likely that the ore-bearing solutions have come in along the steeply dipping faults of the district, but in Bonne Terre some orebodies at least were formed by solutions which moved some distance away from the faults before depositing their load, and the relation of the dips of the rocks was of more importance in the final localization of ore than the faults. The exact percentage of effect exercised by each of these two areally localizing causes could only be determined by careful and detailed mapping.

In two districts in New Mexico[41] the individual orebodies are described as being in local small arches. One of these, the Magdalena district, has been mentioned; the other is the Kingston district.

At Tombstone, Ariz.,[42] Church finds that "the bedded deposits lie in the anticlinals, sometimes on the flank, sometimes in the apex; but the

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[38] W. H. Emmons: Sulphide Ores of the Mississippi Valley. *Econ. Geol.* (1929) **24,** 221–271.

[39] See Fig. 4, taken from E. R. Buckley: Geology of the Disseminated Lead Deposits of St. Francois and Washington Counties, Missouri. Missouri Bur. Geol. and Mines (1909) **9.**

[40] Thanks are due to Mr. C. H. Crane, President of the St. Joseph Lead Co., for permission to use these personal observations, and to Mr. C. K. Hitchcock and Mr. F. Jones of the mine's staff for courtesies extended during the author's visit at the mines.

[41] W. Lindgren, L. C. Graton and C. H. Gordon: *Op. cit.*, 254, 270.

[42] J. A. Church: The Tombstone, Arizona, Mining District. *Trans.* A. I. M. E. (1903) **33,** 3–37.

synclinals are barren." Orebodies in fissures are found chiefly at the intersection of the fissure with an anticline. See Figs. 5 and 6, which illustrate both types.

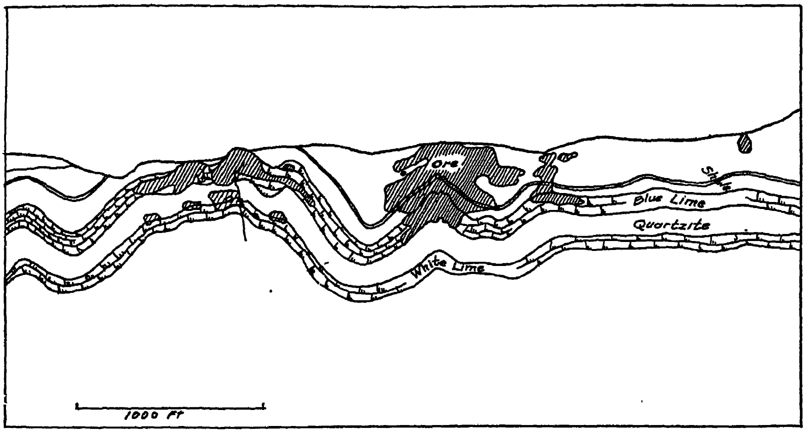

FIG. 5.—SECTION ALONG WEST-SIDE VEIN, TOMBSTONE, ARIZ.
Two types of relationship of ore to anticlines exhibited. Large orebody in vein on one anticlinal crest; other main orebody a replacement of limestone on a crest. Taken from Church.

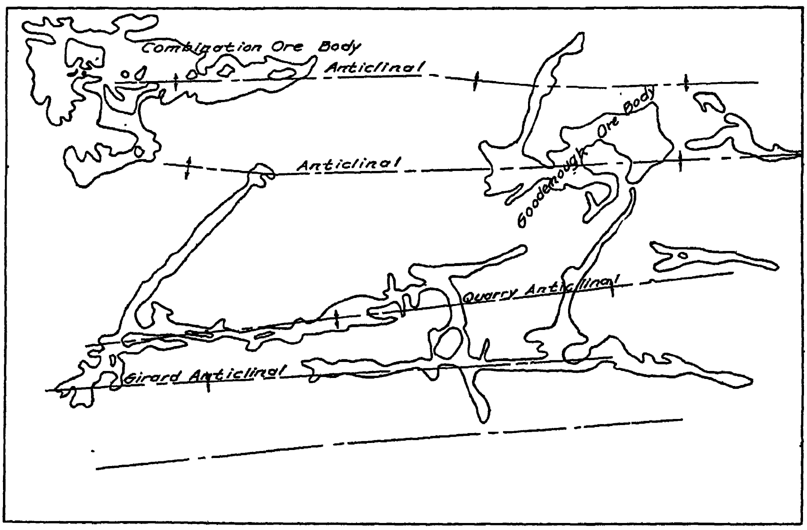

FIG. 6.—HORIZONTAL PLAN SHOWING RELATIONS OF OREBODIES TO ANTICLINES, TOMBSTONE, ARIZ.
Orebodies extending northeast-southwest related to faults. Taken from Church.

In the Lardeau map-area, British Columbia, Gunning[43] finds that the galena-sphalerite deposits in limestone at the Wigwam property show the

[43] H. C. Gunning: Mineral Deposits of the Lardeau Map-Area, British Columbia. Canada Geol. Survey *Mem.* 161 (1929) 25.

following features: "The limestone bed dips on the average 23° to the northeast . . . The largest bodies of sulphides are found where the limestone is flatter than usual or where it is folded into small anticlines. Where it maintains an average or steeper dip little or no ore has been discovered, although silicification exists."

In the Ainsworth mining camp,[44] there is also apparently some relation to terraces. It is stated that "From observations over a limited field it is suggested that the orebodies occur associated with rolls or changes of dip in the surrounding quartzites, the most favorable locality being the areas of low dip which are preceded or followed by areas of high dip."

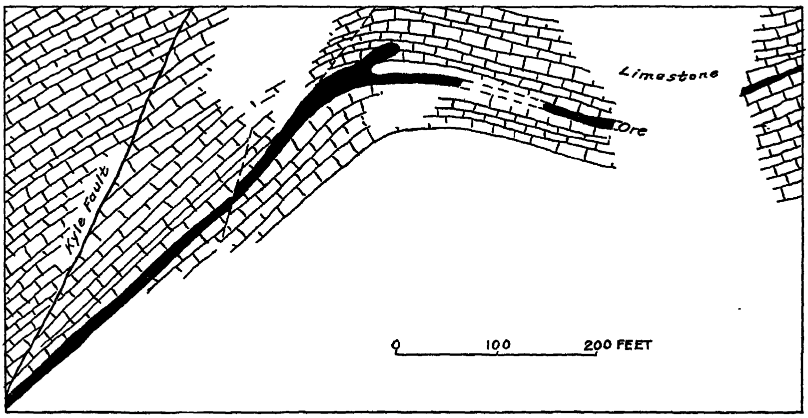

FIG. 7.—VERTICAL EAST-WEST SECTION, HOPE MINE, PHILIPSBURG, MONT.

In the Windemere map-area, British Columbia[45] the "mineralization . . . is associated with minor closed and fractured anticlinal folds."

According to Cooke,[46] the orebodies at the Amulet mine in Quebec are related to crests of anticlines in the lavas which the ore replaces. He states that J. J. O'Neill finds similar structural conditions in the Pend Oreille district, Washington, where the lead-zinc orebodies occupy anticlines in the limestone.

The anticline just east of Philipsburg, Mont., has minor anticlines containing oreshoots[47] (Fig. 7). An interesting group of deposits showing orebodies developed in anticlinal folds, in pre-Cambrian or Paleozoic

[44] S. J. Schofield: Geology and Ore Deposits of Ainsworth Mining Camp, British Columbia. Canada Geol. Survey *Mem.* 117 (1920) 44.

[45] J. F. Walker: Geology and Mineral Deposits of Windermere Map-area, British Columbia. Canada Geol. Survey *Mem.* 148 (1926) 40.

[46] H. C. Cooke: The Amulet Mine, Quebec. *Canadian Min. & Met. Bull.* 219 (1930) 907–914.

[47] W. H. Emmons and F. C. Calkins: Geology and Ore Deposits of the Philipsburg Quadrangle, Montana. U. S. Geol. Survey *Prof. Paper* 78 (1913) 213–219 and map.

rocks are certain of the orebodies of Broken Hill, Australia,[48] Ducktown, Tenn.,[49] and the Sherritt Gordon, Manitoba.[50] Shearing and drag folds are present (Fig. 8). The Homestake mine, South Dakota,[51] is also on a

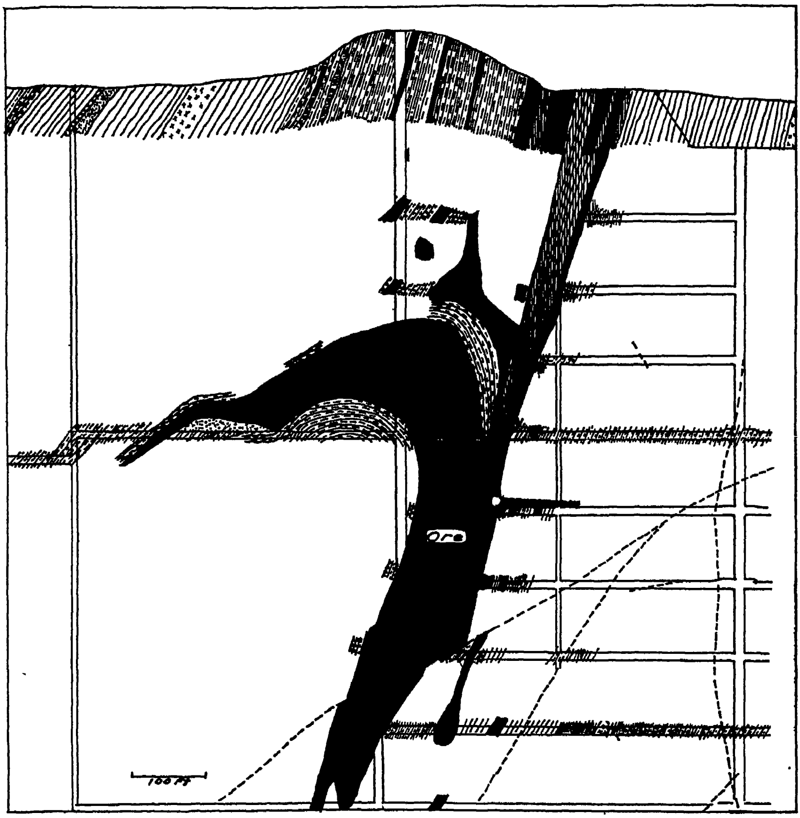

FIG. 8.—VERTICAL SECTION LOOKING NORTH, MAIN BROKEN HILL LODE, BROKEN HILL, AUSTRALIA. ORE REPLACING ANTICLINAL DRAG FOLD.

dome of pre-Cambrian rocks. The details of relationship of ore to minor folds at Homestake are of interest, since one large orebody, the Incline, is on a minor pitching syncline, which has been greatly broken by a porphyry intrusion. Viewed broadly, much of the ore at Homestake

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[48] E. C. Andrews: The Geology of the Broken Hill District. New South Wales Geol. Survey *Mem.* 8 (1922).

[49] W. H. Emmons and F. B. Laney: Geology and Ore Deposits of the Ducktown Mining District, Tenn. U. S. Geol. Survey *Prof. Paper* 139 (1926).

[50] J. F. Wright: Kississing Lake Area, Manitoba. Canada Geol. Survey *Summary Rept.* (1928) B, 73–105.

[51] S. Paige: Geology of the Region Around Lead, S. Dak., and Its Bearing on the Homestake Ore Body. U. S. Geol. Survey *Bull.* 765 (1924).

appears to be related to subordinate pitching anticlines on a nose of the main dome.

The Clover Leaf mine, a few miles north of the Homestake, is also on an anticline with the thickest portion of the orebody at the crest. The country rocks are schists, slates, amphibolites and quartzites. The ore was native gold-bearing quartz with heavy pyrite and some galena.[52]

A variation in type of deposit is that shown by the pyritic deposits of Sulitjelma, Vaddasgaissa and Skorovos, Norway,[53] which are related to anticlinal folds. In part the relation appears to be to a large anticlinal warping, such as was considered in the first part of this paper, but also in part to a smaller anticline at Skorovos.[54] The sections given in this last paper suggest that the structure enclosing the ore is an anticline, although it is modified by a lesser syncline within a part of the orebody.

Other massive pyritic deposits related to anticlines are found at Bully Hill, California,[55] the ore being localized along shear zones at the crest of an anticline, the Bluff orebody at Britannia, B.C., also in an anticline,[56] and it is quite possible that the whole zone at Britannia is along a sheared anticlinal structure.

In the Iwa[57] and Besshi copper mines in Japan, the ore is found in shear zones along anticlinal crests.

At the Pilgrims Rest gold field in South Africa,[58] steeply dipping fractures and anticlinal folds are said to have acted as barriers to movement of solution along bedding planes and thus determined the position of the orebodies. In the Yerranderie silver field in New South Wales,[59] the ore is in fissures and in saddles and rolls in lava and tuff beds.

The metasomatic siderite deposits replacing limestone mostly have a pronounced relation to dome structures. Bilbao, Spain,[60] is on a dome. The siderite deposits which form the main part of the northern

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[52] J. D. Irving: Economic Resources of the Northern Black Hills. U. S. Geol. Survey *Prof. Paper* 26 (1904) 92–94 and map opposite 212.

[53] D. W. Bischopp: The Tectonic Geology of the Sulitjelma Pyrite Deposits in Relation to Their Genesis. *Trans.* Inst. Min. and Met. (1924) **33**, 136–146.

[54] H. H. Smith: Note on the Skorovos Iron Pyrites Deposit, Norway. *Trans.* Inst. Min. and Met. (1922) **32**, 35–47.

[55] A. C. Boyle: The Geology and Ore Deposits of the Bully Hill Mining District, California. *Trans.* A. I. M. E. (1914) **48**, 67–117. See Fig. 1.

[56] H. T. James: Britannia Beach Map-area, British Columbia. Canada Geol. Survey *Mem.* 158 (1929) 94–99.

[57] S. Yehara: Geologic and Tectonic Study of Shikoku. Japanese *Jnl. Geol. and Geog.* (Oct., 1929) **7** [1], 1–42, map.

[58] L. Reinecke and W. G. A. Stein: Orebodies of the Pilgrims Rest Gold Field (Eastern Transvaal). *Trans.* Geol. Soc. South Africa (1930) **32**, 65–88.

[59] L. F. Harper: The Yerranderie Silver Field. New South Wales Geol. Survey, *Min. Resources* 35 (1930).

[60] F. Beyschlag, P. Krusch und J. H. L. Vogt: *Op. cit.*, 494–502.

Africa iron ores[61] are on several domes, and another in the Hüttenberger Erzberg[62] and the siderite veins of Siegerland are on anticlines.[63] Crystalline magnesite deposits replacing limestone are also found related to anticlinal structures, as has been remarked by Bain[64] for deposits in Argentuil County, Quebec, and Stevens County, Washington.

The gold quartz veins in the saddle reefs of Bendigo, Australia,[65] have often been described, and mention should be made of somewhat similar relations in the gold deposits of Nova Scotia.[66] The quartz veins at Bendigo frequently extended down into the synclines, but the gold was found almost exclusively in the anticlinal or saddle portion. Twenty-four "saddle reefs" were found in one mine, one below another on an anticlinal crest. Similar "saddle reefs" are described in the Castlemain gold field,[67] while in the Wood's Point[68] field the gold is in the main closely related to igneous dikes which "occur more frequently near the disrupted beds of an anticline than elsewhere."

The El Callao vein in Venezuela,[69] which produced upwards of thirty million dollars, was mainly along the bedding planes of the rocks, on the flank of a fold, and in the lower synclinal portion the quartz continued but the gold played out.

The sulfur deposits of the Louisiana-Texas field, as is well known, are found on domes. The sulfur in the limestone has definitely been introduced after fracturing.[70] Of other commercial sulfur deposits in the United States, one with hot spring deposits is found at Thermopolis, Wyo.,[71] on an anticline, and another also with hot spring deposits has been mined near Cody, Wyo.,[72] on a terrace of an anticline.

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[61] P. Geijer: Geological Relations of the North African Iron Ores. *Econ. Geol.* (1927) **22,** 537–564.

[62] F. Beyschlag, P. Krusch und J. H. L. Vogt: *Op. cit.* 488–490.

[63] H. Quiring: Antiklinal Erzmäntel im Siegerland. *Metall u. Erz* (1928) **25,** 519–525.

[64] G. W. Bain: Magnesite Deposits and Their Origin. *Econ. Geol.* (1924) **19,** 403.

[65] E. J. Dunn: Reports on the Bendigo Gold Field, Nos. 1 and 2. *Special Reports,* Dept. Mines, Victoria, Australia (1896) 15–25.

[66] E. R. Faribault and W. Malcolm: Gold Fields of Nova Scotia. Canada Geol. Survey *Mem.* 156 (1929).

[67] W. Baragwanath: The Castlemain Gold Field. Victoria Dept. Mines *Mem.* 2 (1903) 11–32.

[68] O. A. L. Whitelaw: The Wood's Point Goldfield. Victoria Dept. Mines *Mem.* 3 (1905) 11–12.

[69] W. H. Newhouse and G. Zuloaga: Gold Deposits of the Guayana Highlands, Venezuela. *Econ. Geol.* (1929) **24,** 797–810. See literature cited.

[70] W. Lindgren: Oral communication.

[71] E. G. Woodruff: Sulphur Deposits near Thermopolis, Wyo. U. S. Geol Survey, *Bull.* 380 (1908) 373–380.

[72] E. G. Woodruff: Sulphur Deposits at Cody, Wyo. U. S. Geol. Survey *Bull.* 340 (1907) 451–456.

In connection with these last two sulfur occurrences, which are with hot spring deposits, it may be pointed out that hot springs may be localized along faulted anticlines,[73] such as the Hunters Hot Springs, Montana, and Hot Springs, Arkansas[74] the latter being on the southwest end of an anticline which pitches steeply southwest. Most of the ninety thermal springs in northwest Virginia and adjacent parts of West Virginia issue on anticlines, according to Reeves.

A type of considerable interest is where ore is localized along a cross fault or fissure zone at its intersection with an anticline. The relations in the veins at Chañarcillo, Chile,[75] show this as well as other interesting features. The following quotation is to the point: "The richest vein of Chañarcillo, the Corrido Colorado, continuous over a distance of more than two kilometers, lies directly upon the crest of the greater anticline. The Veta Descubridora, a rich producer, is upon the axis of the minor, divergent anticline. Away from the axes of the folds the veins parallel to them grow less continuous and poorer. Among the veins at 40° to the anticlinal axes, but one, the Veta Candelaria, is rich, and attains its best development upon the crest of the two folds at their junction."

Lead and zinc ores in faults which carry ore mainly where cutting anticlines are described in the area between the Ruhr and Rhine.[76] The West side vein at Tombstone, Ariz.[77] (Fig. 5), and the orebody of the Drumlummon mine, Montana,[78] are other examples.

The Selbecker vein[79] and others on the Velbert anticline show similar features, as do also numerous veins between Clausthal and Lautenthal in Germany.

The tendency toward upward movement of certain mineralized constituents is also indicated by the occurrence of tin at the apexes of domes of pegmatite and on the hanging walls of flat dipping dikes and sills in eastern Manitoba,[80] and the cryolite mass at Ivigtut, Greenland[81] is with

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[73] U. S. Geol. Survey *Bull.* 260 (1905) 601–604.

[74] A. H. Purdue and H. D. Miser: Hot Springs Folio, Arkansas. U. S. Geol. Survey *Geol. Folio* 215 (1923) 11.

[75] W. L. Whitehead: The Veins of Chañarcillo, Chile. *Econ. Geol.* (1919) **14** 02,.

[76] F. Unterhossel: Empfiehet sich eine erneute Untersuchung der Selbeck-Lintorfer Grubenfelder ausserhalb der durch den Bergbau erschlossenen Feldesteile? *Metall u. Erz* (1928) **25**, 242–245.

[77] J. A. Church: *Op. cit.*

[78] C. W. Goodale: The Drumlummon Mine, Marysville, Montana. *Trans.* A. I. M. E. (1914) **49**, 258–283.

[79] A. Stahl: Über die Beziehungen der Erzführung einiger Blei-Zinkerzgänge zur Tektonik des Nebengesteins. *Ztsch. f. prakt. Geol.* (1920) **28**, 12–14; 28–34.

[80] D. R. Derry: Tin-bearing Pegmatites in Eastern Manitoba. *Econ. Geol.* (1930) **25**, 145–159.

[81] R. Baldauf: Über das Kryolith-Vorkommen in Grönland. *Ztsch. f. prakt. Geol.* (1910) **18**, 432–446.

pegmatite in the axis of a pitching anticline in the gneiss. The tendency is also seen for lead and zinc oreshoots to be on terraces in upward convexities under porphyry sills at Leadville, Colo.[82]

RELATIONS OF SYNCLINES TO ORES FORMED BY ASCENDING SOLUTIONS

One of the striking features of this relatively small group is the fact that most of them depart from the more normal type of ore deposits in other features than the structural relation. In general, their origin has been and still is a debatable subject.

The ore deposits of Franklin Furnace, N. J., of Meggen, and of Mansfeld, Germany, lie in synclines. Some of these may be of sedimentary origin, and since the group throws little light on the subject we are considering they will not be discussed further.

The magnetite iron ores of New York are in both anticlines and synclines, as are also some of those in Sweden, although some show no pronounced relationship to either type of fold.

By grouping different types of the clearly epigenetic deposits found in synclines, some interesting relationships are found. The majority are in gently folded rocks, the Mississippi Valley type of zinc deposits being typical; to these may be added those of Cobalt, Ont., and Kennecott, Alaska. This group will be considered at some length in a later paper because there are some relations between the large and small units of structure involved which can be made clear for several districts only by maps, sections and full discussion.

It might be expected that where rocks are strongly folded, faulting with ensuing ore deposition would take place in synclines. Few examples have been found by the writer, but in the Tsumeb mine, South Africa,[83] the ore is along a fault in steeply folded rocks near the base of a small syncline. Other examples are at Idria, Italy,[84] and lead-silver zinc veins in Pribram, Bohemia,[85] which traverse the flank and with depth the axial portion of an overturned syncline. The gold deposits at Kirkland Lake, Ont., may also be mentioned.

The two oreshoots at Tyee, B.C., are on the flanks of a strongly folded syncline.[86,87] The deformation was probably so intense at these places that faulting or openings were not localized by the anticlines.

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[82] S. F. Emmons, J. D. Irving and G. F. Loughlin: Geology and Ore Deposits of Leadville Mining District, Colorado. U. S. Geol. Survey *Prof. Paper* 148 (1927). See sections of Downtown district opposite 32, and of Fryer Hill opposite 280.

[83] H. Schneiderhöhn: Das Otavi-Bergland und seine Erzlagerstätten. *Ztsch. f. prakt. Geol.* (1929) **37**, 85–116.

[84] L. de Launay: *Op. cit.*, **3**, 437.

[85] F. Beyschlag, P. Krusch und J. H. L. Vogt: *Op. cit.*, 229–235.

[86] C. H. Clapp: Sooke and Duncan Map-Areas, Vancouver Island. Canada Geol. Survey *Mem.* 96 (1917) 389.

[87] W. H. Weed: Notes on the Tyee Copper Mine. *Eng. & Min. Jnl.* (1908) **85**, 199–201.

Where mineralization is intense beds on both anticlines and synclines may show replacement by ore. It is probable that the mineralization is strong enough to attack the easily replaceable bed irrespective of structural attitude. At the Pinnacles mine,[88] near Broken Hill, Australia, certain beds in the schists were replaced on both anticlines and synclines. The ore is thicker in the crests of the anticlines and in the synclinal troughs than on the flanks of the folds. In part this thickening is probably due to the thickening of the replaced bed. Two deposits of contact metamorphic type with low-grade copper ores are described as being in limestone basins, or synclines—at Phoenix, Boundary district, B.C.,[89] and at the Franklin mining camp, B.C.[90]

The North Star mine, near Kimberley, B.C., formerly worked two orebodies in two different synclines which were separated by an anticline. Schofield[91] states that "The orebodies probably represent remnants of a once continuous orebody, the larger part of which has been removed by erosion."

The Rhodesian copper deposits are in synclines.[92] They also appear to be uniform and persistent replacements of certain stratigraphic horizons under impermeable beds. Erosion has removed the intervening anticlinal areas, but in the Katanga[93] region of the Belgian Congo, with ore in dolomites in the same stratigraphic series and where erosion has not been so deep as in the Rhodesian belt, the ore deposits are found along faults and in anticlines. In the Rhodesian field, after full exploration is made, it will be of interest to compare the value and extent of the ore obtained along the trough line of the synclines with that of the ore on the flanks.

Theoretical Considerations

The idea might be advanced that large anticlines more frequently form stream divides with good rock exposures while the synclines are in

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[88] W. J. Turner: On the Geology of the Pinnacles Mine and District. *Proc.* Australasian Inst. Min. and Met. (1927) **68**, 299–312.

[89] O. E. LeRoy: The Geology and Ore Deposits of Phoenix Boundary District, B.C. Canada Geol. Survey *Mem.* 21 (1912) 53–58 and map.

[90] C. W. Drysdale: Geology of Franklin Mining Camp, B.C. Canada Geol. Survey *Mem.* 56 (1915) 166.

[91] S. J. Schofield: Geology of Cranbrook Map-Area, B.C. Canada Geol. Survey *Mem.* 76 (1915) 134.

[92] A. M. Bateman: The Rhodesian Copper Deposits. Canadian *Min. & Met. Bull.* 216 (1930) 477–513.

A. Gray: The Correlation of the Ore-bearing Sediments of the Katanga and Rhodesian Copper Belt. *Econ. Geol.* (1930) **25**, 783–804.

[93] P. Kovaloff: Copper Deposits of the Watershed of the Congo and Zambesi Rivers. *Min. & Ind. Mag. of South Africa* (1930) **10** (3), 124–125.

V. G. Douglas: Observations on the Geology and Mines of the Belgian Congo. *Min. Mag.* (1930) **42**, 337–348.

valleys and buried by debris, the ore deposits consequently being found largely in the anticlines. Indeed, the explanation has been used in part for the association in Mexico.[94]

This particular adjustment of structure and topography, such as is found to perfection in the young Jura Mountains,[95] is rare, since viewed broadly ore deposits are found in regions showing various stages of topographic development or adjustment of topography to structure.

Several structural causes may be suggested as localizing epigenetic ore deposits on the flanks of folds or on the crests of anticlines:

1. Source of solutions in intrusive rocks, such as stocks which are underneath the upwarped area.
2. Differences in faulting on upwarps as compared with downwarps. These differences are of two kinds. More faulting on upwarps than in synclines and more openings along the faults or fractures on the upwarps.
3. Gathering of solutions by beds and arches with a possible final collection in the uppermost part of the structure.
4. Impounding or damming of solutions.

The first two would apply chiefly to the larger structures, while the last two apply mainly to smaller ones.

Intrusive igneous rocks have not been found with all anticlines containing ore. The frequent presence of intrusive rocks associated with the upwarps has caused some geologists to believe that all are so related although erosion or mine workings may not have revealed them in some of the mining districts.

Spurr[96] has discussed at some length the doming of certain mineral districts. He demonstrates in some places an obviously close relation between the intrusion and the doming of older rocks above it. Domes without visible intrusive rocks, such as the one on which the southeastern Missouri lead deposits are located, are also considered by him to be due to the intrusion of underlying igneous rocks. He concludes in part that "not only doming, but folding and faulting—even overthrown folds and overthrust faults—may be due to magma migration."[97]

The author thinks it is still an open question as to whether stocks of intrusive rocks force up the older rocks arched over them, or whether both the upwarp and the intrusive are due to the same causes. It is difficult to conceive of competent folding producing the domes such as Spurr has described, and it seems necessary to assume to some degree an upward thrust by the intrusive magma, which is now represented by a stock or

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[94] A. R. Fletcher: *Op. cit.*

[95] W. M. Davis: Geographical Essays, 453–454. 1909.

[96] J. E. Spurr: Ore Magmas, **1**, 187–252. New York, 1923. McGraw-Hill Book Co.

[97] J. E. Spurr: *Op. cit.*, 187.

batholith. Following the usage of Willis,[98] competent folds are defined as due to initial horizontal compression, in the plane of the beds; incompetent folds to vertically acting forces normal to the beds.

Also, the numerous examples of the collapsed arch type of faulting in mineralized districts, which has been emphasized by Spurr,[99] would seem to show that the folding is not entirely competent and that therefore the intrusion did exert some vertical force in the upwarping, and when this support is relaxed by shrinkage of the intrusion during crystallization of the magma, faulting due to gravity takes place.

It might be considered that this is the same as saying that areas of igneous intrusion show much faulting. It may be suggested, however, that in a district of intrusion those portions of the older rocks which are upwarped with intrusives beneath are affected by more vertically acting stresses both during and following intrusion than the near-by synclines within the general intrusive area, and therefore are more faulted. As Willis points out,[100] there would be the tendency toward tension in the beds of such an incompetent anticline during intrusion.

Some of the minor folds at Bonne Terre, Mo., are the result of initial dip of the sediments plus probably slight settling over buried ridges of an old topography. Nevin and Sherill[101] have suggested the term "compaction fold" for this type of structure. Such folds are incompetent with the acting stresses, being essentially vertical. The resulting slight local folding with some fracturing around the peaks of the buried topography would probably be more pronounced than in intervening synclines. This type probably is not very common.

In competent folding, where the folds are of fairly large dimensions, analysis suggests certain differences in the stress relations of the anticlines as compared with the synclines. After folding, and when static conditions are reached, the competent beds will act in part as struts transmitting part of the weight of the anticlinal structure into the syncline. In general, also, more material is vertically above the syncline than the anticline. Both these features would tend to cause higher rock pressures in the syncline. Of the two types of folds, then, fractures or faults would have more tendency to remain open on the anticlines.

In folding by horizontally acting forces there would probably be more relief of excessive stresses by fracturing and faulting near the crests of large anticlines than in the troughs of the corresponding synclines.

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[98] B. Willis and R. Willis: Geologic Structures, 246–261. New York, 1929. McGraw-Hill Book Co.

[99] J. E. Spurr: The Relation of Ore Deposition to Faulting. *Econ. Geol.* (1916) **11**, 601–622.

[100] B. Willis and R. Willis: *Op. cit.*

[101] C. M. Nevin and R. E. Sherrill: Studies in Differential Compaction. *Bull. Amer. Assn. Petr. Geol.* (1929) **13**, 1–22.

Horizontally acting stresses may produce folding, which as the severity increases may pass into thrust or reverse faults, often with associated drag folds. Such thrust faults or shear zones, usually of high dip where ore deposits are found, probably will be largely confined to anticlines.

Broken Hill, Australia, and the Beshi and Iwa copper deposits in Japan, the Bawdwin mines, Burma, and probably Ducktown, Tenn., and Sherritt Gordon, Manitoba, may be given as examples.

In review it may be stated, then, that there are some grounds for believing that under certain conditions anticlines may be more faulted or fractured than synclines. There also exist grounds for believing that with fairly large structures more openings would be present on the anticlines, that there would be less pressure holding broken rock surfaces together. More fractures and those which are present being in a state of "potential openings" would decidedly favor ore deposition.

The gathering of solutions by permeable beds, or under impermeable covers which converge upward, to form an anticlinal-like crest, or which may even converge quaquaversally, is, the writer believes, of much importance.

This factor probably is more active in the smaller structures than in the ones previously considered. There appears to be a pronounced tendency for the solutions to rise within a structure. The writer has been very forcibly impressed by the analogy of the occurrence with that of petroleum in somewhat similar structures. This has been remarked before by Stahl.[102]

Damming of Solutions

The structures are such as would suggest that ores are frequently deposited where solutions are dammed. Some of the examples which are given as showing this feature are doubtless due in part to fracturing localized underneath the impermeable cover; the fractured rock being replaced. Review of the occurrences described, and of many others in the literature, supports this view for numerous examples, but others show features suggesting that the impounding of rising solutions in a dome, or on a terrace, has caused ore deposition.

In the case of descending solutions that are dammed by impervious troughs, or synclines, the solutions probably are in large part annually periodic, which would result in the basin containing more solution for a longer time than any other adjacent structure.

Physical-chemical considerations suggest that increased time for reaction where constantly rising solutions are dammed is not the main factor that causes the deposition of an oreshoot. This is dependent on

[102] A. Stahl: Über die Beziehungen der Erzführung einiger Blei-Zinkerzgange zur Tektonik des Nebengesteins. *Ztsch. f. prakt. Geol.* (1920) 12–14; 28–34.

the fact that as the solutions spread, and the time for reaction increases, the rate of flow will decrease in approximately the same proportion. The writer believes that the principal cause, although others contribute, is that more replaceable rock is saturated by ore-bearing solutions at this point than elsewhere. If the rock is a competent limestone bed underneath shale, there would also be a tendency towards fracturing on the upper or outer part of the bend in the limestone adjacent to the shale (Fig. 3). Such fractured material would form an excellent locus for ore.

Summary

Epigenetic ore districts on folded rocks are nearly always on the flanks or crest of an anticlinal fold. Few districts are on the trough line of a syncline. Individual oreshoots also show the relations just mentioned. The individual oreshoots are frequently in small anticlines or terraces superimposed on larger anticlinal structures. The few epigenetic oreshoots which are found in synclines are mainly in three classes: (1) where deformation has been strong, (2) where mineralization has been thoroughgoing, in which case both synclines and anticlines may be mineralized, and (3) in gently folded rocks. The third class will be considered in a later paper. It is the only group of the three in which the synclinal structure appears in some districts to have been instrumental in localizing the ore; in the other two the deformation or mineralization has been so thoroughgoing or intense that the effect of fold structure on localizing ore was nullified.

Theoretical Conclusions

Several features cause epigenetic ore deposits to form in anticlinal areas. No one cause can be assigned. In some examples the igneous rock which furnished the ore-bearing solutions is underneath the upwarp. There are good theoretical considerations, which indicate that faulting or fracturing may be localized by anticlines in certain types of folding. Such faults would serve as conduits for the ore-carrying solutions. Less rock pressure is believed to be present on the crests of some types of anticlines than in adjoining synclines. More openings or more potential openings would thus be available in the fractured rocks. Solutions may be gathered into an inverted trough, or in the crest of a dome by permeable beds converging upward, or by impermeable roofs converging upward.

Damming of solutions in an anticlinal crest, a dome or on a terrace, favors the formation of oreshoots by increasing the volume of rock saturated by ore-bearing solutions.

Lastly, the observations are interpreted to mean that the ore-bearing solutions have a strong tendency to rise to the higher parts of a permeable structure. Probably this means they move toward the areas of lower pressure.

ACKNOWLEDGMENTS

Thanks are due to Messrs. W. Lindgren, E. Cohen, F. K. Morris and M. J. Buerger for advice and helpful discussion.

DISCUSSION

(Albert O. Hayes presiding)

A. O. HAYES, New Brunswick, N. J.—The wealth of data now scattered in separate papers needs assembling and the work which Dr. Newhouse has accomplished is most welcome. While we may not wish to go as far as he has in his view of the general importance of anticlines for orebodies, his paper presents a point of view which we are glad to discuss.

I. B. JORALEMON, H. A. BUEHLER, A. M. BATEMAN, D. H. MCLAUGHLIN AND R. D. HOFFMAN contended in the main that general conclusions could not be drawn as the question was yet too broad to be proved by the evidence presented; that orebodies are found on tops of anticlines because they are more frequently exposed while synclines are hidden; and W. T. THOM, JR., pointed out that several of the so-called anticlinal structures are uplifts due to faulting or warping, and are not anticlines in the usual sense of the word. W. H. NEWHOUSE replied that the evidence was too conclusive to be disregarded and that in his opinion the result of geologic study by a great number of men all over the world is more to be depended upon than any one man's personal observation.

E. D. WILSON, Tucson, Ariz. (written discussion).—In accord with the general ideas set forth in Dr. Newhouse's article, an additional example of the relation of hypogene mineral deposits to structural features is offered by the unique chrysotile asbestos deposits of Arizona.[103]

Chrysotile, which is the asbestiform variety of the hydrous magnesium silicate, serpentine, here occurs as cross fiber veins in the Mescal dolomitic limestone, near intrusive contacts of diabase. The asbestos-bearing limestone in general is nearly horizontal, but, in detail, displays considerable local tilting and small-scale folding that apparently accompanied the diabase intrusion. In places, a minor amount of normal faulting has caused displacements of a few feet to a few hundred feet magnitude. Faults of very small throw, and several systems of visible fractures, are rather numerous. Part of the fracturing and faulting is earlier than the asbestos and part is later.

Asbestos-bearing bodies are apt to occur where a zone, or two interlacing zones, of fractures cut across a Mescal limestone that is favorable in composition and favorably situated in reference to the diabase. This fact has been used for some years by the larger mines in prospecting, and has been recorded by Trischka.[104] Where a minor fold happens to be in the plane of such an intersection of fractures, the conditions for an asbestos deposit are especially favorable. Furthermore, good fiber occurs along the crests, troughs or flanks of many minor folds that have not been appreciably fractured, particularly if they are to one side of a transverse diabase contact.

These relationships are well defined at the Arizona Asbestos Association deposit, in which is situated the largest asbestos mine of the United States. A low, gentle anticline, modified by some transverse warping, arches over the area in which most

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[103] E. D. Wilson: Asbestos Deposits of Arizona; With an Introduction on Asbestos Minerals by G. M. Butler. Ariz. Bur. Mines *Bull.* 126 (1928).

[104] C. Trischka: Asbestos and the Arizona Industry. *Eng. & Min. Jnl.* (1927) **124**, 337–340.

of the asbestos of the American Ores mine occurred. The most productive area in the Regal mine occupied a low, crenelated, structural dome.

P. C. Benedict, Jerome, Ariz. (written discussion).—Dr. Newhouse has indicated that a few ore deposits occur in synclines, many on the flanks of anticlines, and a goodly number on anticlines. The flank of an anticline is the flank of a syncline, hence flank occurrences are stripped of much of their significance. However, such "flank" deposits would obtain much more importance were exact data available that deposits occurring nearer the axes of anticlines predominate in pounds of metal over those situated nearer the axes of synclines. I believe this to be a fact for most types of deposits. Dr. Newhouse doubtless is of the same opinion, but the data from which he worked were often incomplete as to the quantitative distribution of ore, and he has been too honest to emphasize a relation not positively warranted by the available data.

Two examples of anticlinal orebodies not given by Dr. Newhouse are briefly described below:

At the San Carlos mine, Chihuahua, Mexico, 40 miles southeasterly from Presidio, Tex., the orebody, in limestone, is elongated parallel with the axis of an anticline, and the center of the orebody is about 1000 ft. east of the axis. The exact position of the syncline to the east is hidden under a capping of volcanic rocks, but is at least two miles distant. The dips, east of the axis, are relatively flat to almost the easterly edge of the orebody, beyond which they steepen considerably. The orebody is about 600 ft. wide, which brings its western edge within 700 ft. of the anticlinal axis.

Furthermore, the ore does not favor any one horizon, but occurs as a dome of mineralization cutting through the beds. Mineralization ridges on top of the mineralization dome show the best ore; mineralization valleys are poorer to uncommercial. San Carlos is a contact metamorphic lead-silver deposit.

At Swansea, Yuma County, Ariz., the known commercial ore occurs in two structural positions; first, along the Swansea fault, and having a flat rake to the northeast; second, according to mapping by F. E. Calkins, along the axis of an anticline plunging flatly to the northeast, which occurs a short distance northwest of the Swansea fault. Most of the anticlinal ore occurs in limestone along its contact with underlying schist. The rake of the ore along the fault is parallel to the plunge of the anticline. Swansea is a copper deposit, probably of contact metamorphic origin.

I particularly favor Dr. Newhouse's statements: "Less rock pressure is believed to be present on the crests of some types of anticlines . . . More openings, or potential openings, would thus be available in the fractured rocks . . . ore-bearing solutions have a strong tendency to rise to the higher parts of a permeable structure . . . towards areas of lower pressure."

Some of the discussers seem to assume that Dr. Newhouse contends that he has the one and final solution for finding ore. I cannot find this attitude in the paper. Many of our most active ore hunters have emphasized the impossibility of formulating rule-of-thumb methods for finding ore. Geological phenomena have not been reduced to mathematical formula. A development recommendation based on geological reasoning is never absolute, and should always be based upon an attempt to weigh chances and expense involved against probable profit.

Dr. Newhouse has been criticized for drawing conclusions partly from published descriptions. Must he spend the remainder of his life reexamining the ore deposits in folded rocks before he is justified in summarizing an important condition?

I wish to express my appreciation to Dr. Newhouse for the considerable amount of work that he has expended to summarize and make available a readily recognizable relationship, which may be classed as another tool, albeit dangerous in unskilled hands, in the incomplete kit of ore hunters.

F. W. SMITH, Redlands, Calif. (written discussion).—Sierra Almagrera, in Almeria province, southeastern Spain, is a mountain 13 km. long, trending northeasterly, by 5 km. wide at widest, rising at its highest to about 350 m. above sea level. The formation is all pre-Cambrian (?) schist; no intrusives were seen. The Mediterranean washes its southeastern base. At its northwestern base are Miocene rocks, not examined but apparently tuffs. The schist plunges beneath this formation; probably there has been some faulting along the contact.

The lead-silver mines, scene of a great boom in 1839–44, were opened on narrow veins of barite and siderite that did not outcrop. The crest of ore deposition was usually less than 100 m. above sea level. At 30 m. above sea level hot water (43° C. and higher) was encountered in considerable volume in 1844; since then the history of the district has been chiefly a tale of woe with Agua Caliente playing the role of villain. At time of visit the mines had been idle most of the time since 1912 and not much could be seen underground.

The district is 5 km. long and 1 km. wide, trending with the axis of the mountain. Within it the schist is a distinct anticline, the schistosity dipping steeply toward the sea on the southeastern side, and in the opposite direction on the northwestern flank. Along the crest of the ridge and in at least one 300-m. shaft sunk near the crest, it is nearly horizontal, dipping at low angles one way or the other. The crest of the anticline is offset northerly on the eastern side by occasional faults, which in depth perhaps are veins.

The number of productive veins is variously stated at 30 to 52. According to old maps, most of them strike between north and northwest and have steep dips. In productive length they were usually 1000 m. or less and the productive portion was that part traversing the anticlinal ridge and the upper part of the northwestern flank. They are reported to have been commercially barren under the seaward slope of the mountain. They are distinct within the explored zone, which at its deepest goes about 200 m. below sea level, but all receive their hot water from one great source, as proved by the results of sinking a pump shaft at the southwestern end of the district and running a gallery 300 m. northeasterly until the first water-bearing fissure was cut, 220 m. below sea level. Pumping then drained all mines at a flat hydraulic gradient for 5 km. northeast.

Some of the veins may be continuous into the Miocene rocks to the northwest, where there are old iron mines, said to have been first worked by the Phœnicians. They were not visited. It was reported that the ore was limonite and siderite, barren of other metals. The rather close coincidence of the productive lead-silver belt with the crest of the anticline is noteworthy.

D. H. BRIEN, Seoul, Chosen (Korea) (written discussion).—All earth movements towards the surface represent a letting up of the pressure at depth; the result is broken or fissured rock, and this, whether plutonic, volcanic or seismic relief from lateral pressure by folding: the anticline. Hence the formation of ore channels along the crests and down the weakened and more or less fractured sides. The same phenomena may occur along a fault fissure, as witness the many cases of a fault-dragged lead, where an enriched orebody is found up to, sometimes, the very wall of the fault or dike, and when the lead is found on the opposite side it is often a tightly closed material even in the drag, and barren.

The shaken rock from any cause, folding, faulting, eruption, settling sometimes, will provide, if within reach of the great solvent water, the "channel of least resistance" for the flow, with or without pressure, of acid, alkaline, any sort of fluids, and if helped by more or less impervious silex enclosing rock—as in the quartzites of the Coeur d'Alene—probably will cause some electrolytic action to help produce the ions which we at present believe results in the final action of ore production, granted the near presence of certain primal elements or their compounds.

Because of these known facts, I believe the author—who, I infer, believes that the majority of oreshoots are connected with anticlines—should have reasoned from the primary cause of the deposits occurring along the loosened fold tops, and made the matter clearer by considering the *form* name of any shaken rock as of secondary value in the genetics of orebodies; as comparably speaking, gneiss, schist, shale, slate, etc., are only changed secondary or tertiary *forms* in the genesis of rocks, and geologically and mineralogically are not rocks; so anticlines are but one form of the many known "shake" structures which have secreted ores under certain known conditions.

I plead for the retention of technical value of terms, by not singling out one class of instances from a list of many nearly similar cases.

R. H. Rastall, Cambridge, England (written discussion).—On page 226, Professor Newhouse refers briefly to the lead-zinc-barite deposits of Shropshire, England, which lie in well-marked anticlines. He continues, "also the most important lead-producing area in Great Britain on the Pennine anticline including the region of Derbyshire, West Yorkshire, and an area to the north."

In 1928 I wrote as follows:[105] "Again in Derbyshire the veins are confined to the well-jointed limestone, and stop off short when they come up against the impervious rocks of the Yoredale facies. This is very noticeable in the small domelike inlier of Ashover, where there are some very large fluorite veins. The general lie of the Derbyshire veins is just as if the limestone had acted as a gasometer, confining the metal-bearing fluids and depositing them in the open fissures, especially near its top." Both here and farther north there is also a good deal of replacement, as "flats."

There is also a considerable copper-lead-zinc mineralization with barite, but without fluorite, in the Ordovician rocks of the English lake district, which is notoriously a dome about 40 miles in diameter, and the lead ores of the Mendip Hills, worked, and nearly worked out, by the Romans, lie in a series of strongly folded and overthrust Armorican anticlines, with east-west strike.

In Cornwall the rich tin-wolfram ores lie in cupolas on top of a big granite batholith, but this is another story.

W. A. Richelsen, Kennecott, Alaska (written discussion).—The paper by Mr. Newhouse has summarized many interesting and important points regarding the relation of structure to epigenetic ore deposits, but his classification (pp. 228 and 240) of the types of deposits at Kennecott, Alaska, as occurring in the Permian near the trough of a syncline and in gently folded rocks requires further discussion.

A great deal of additional field work has been done, and also additional information published, since the paper given by Bateman and McLaughlin. In referring to the age of the Nikolai greenstone and the Chitistone limestone Bateman[106] has used the classification of Moffit and Capps.[107] This is practically the same as that made by Martin,[108] who gives the age of the Nikolai greenstone as probably Permian or early Triassic and the Chitistone as Upper Triassic. Considering the age of both formations and the fact that commercial ore occurs only in the lower member of the Chitistone limestone it would not seem advisable to classify the Kennecott deposits as Permian.

The district is characterized by strong folding and faulting, the general axis of the folding having a northwest, southeast strike, indicating that the strongest horizontal compression was in a northeast, southwest direction. Maximum fracturing would therefore occur near the extremities of the flank of a large syncline or as Newhouse suggests (p. 243), near the crests of large anticlines.

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[105] *Geological Magazine* (1928) **65**, 275.

[106] A. M. Bateman and D. H. McLaughlin: *Op. cit.*, 7.

[107] F. H. Moffit and S. R. Capps: U. S. Geol. Survey *Bull.* 448 (1911) 62, 63

[108] G. C. Martin: Mesozoic Stratigraphy of Alaska. U. S. Geol. Survey *Bull.* 776 (1926) 10, 18.

The general opinion now is that the various types of deposits at Kennecott—namely, transverse and longitudinal fissures, sheeted zones, and fault fissures—occur on the comparatively steeply dipping south flank of a large syncline which may be traced for a distance of 30 miles and which has a strike of approximately N. 60 W.

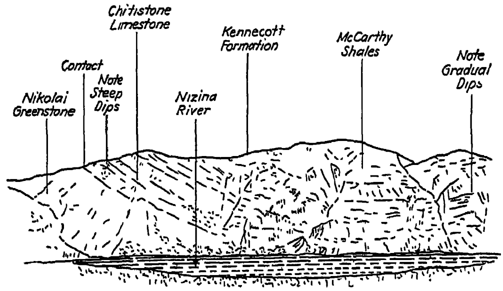

FIG. 9.—GEOLOGY AND SYNCLINAL STRUCTURE OF CHITISTONE LIMESTONE EXPOSED ON WEST SIDE OF NIZINA RIVER, NEAR MOUTH OF CHITISTONE RIVER, ALASKA, LOOKING NORTHWEST. (*Sketched from Plate V, U. S. Geol. Survey Bull.* 448.)

and pitches about 6° to the northwest. The panoramic sketch (Fig. 9) shows the synclinal structure as exposed 8½ miles southeast of Kennecott.

The fact is recognized that Newhouse is referring to the very slight synclinal folding which occasionally occurs on the flank of the large syncline. This is only one of the several theories offered by Bateman[109] in explanation of the origin of fissures. Extensive underground exploration in the past years has shown that the lower beds

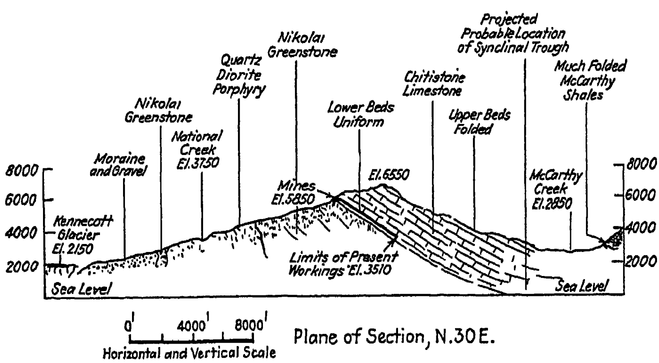

FIG. 10.—CROSS-SECTION IN VICINITY OF MINES AT KENNECOTT, ALASKA.

of the Chitistone limestone in the vicinity of the mines have an average dip of about 33° and also have a surprisingly uniform structure parallel to the bedding; that the slight synclinal folding is difficult to recognize and when prominent is usually associated with drag folding in connection with transverse and bed faulting and also that some of the prominent orebodies of the transverse type show no evidence of synclinal folding. The synclinal theory, consequently, has become of secondary importance and much attention is now being given to the alternate theory offered by Bateman[110] that differential movement of the beds along the bed faults, with or without irregularities, has caused rupture and the attendant fissuring and this may be further

[109] A. M. Bateman and D. H. McLaughlin: *Op. cit.*, 54.
[110] A. M. Bateman and D. H. McLaughlin: *Op. cit.*, 57.

elaborated by including; that the slight irregularities and rolls in the bed faults have been caused by the bed faults following ground weakness rather than definite stratigraphic horizons.

The geologic cross-section in the vicinity of the mines (Fig. 10) shows the location of the workings in reference to the general structure and it is at once evident that the present known deposits occur well up on the flank of the large syncline and several thousand feet above the probable location of the main trough. If considered from the viewpoint of the anticlinal theory they would lie near the crest of the anticline as shown by Bateman in Fig. 3 of his paper.

Considered in a broad general way the Kennecott deposits may be classified as replacement types occurring in the Triassic and closely related to both bed and transverse faulting in the lower beds of the Chitistone limestone and near the extremities of the flank of a large syncline.

W. Lindgren, Cambridge, Mass. (written discussion).—Some years ago[111] I ventured to call attention to the great accumulation of detail in description of ore deposits and to the necessity of critical examination, comparison and coordination of these data. To dig out and to assemble the literature is an arduous, many might think a thankless task, but it is absolutely necessary if the science is not to lose itself in the mazes of descriptive detail. I pointed out that some such work had been done but that a vast amount of work still remained; therefore, I greet with pleasure and appreciation this attempt by Dr. Newhouse to correlate structure and ore deposition. Perhaps in late years we have paid a little too much attention to composition and microscopic structure and given too little time to the study of structural relations, and I think that Dr. Newhouse has performed a laborious task in an excellent manner. The title of the paper is broad enough, and if as a by-product of the work it has been ascertained that anticlinal structure offers a preferred locus for ore deposition, we have made a distinct step in advance. Doubtless there is more than one explanation of this relationship as well explained by the author. It is easy to criticize such a contribution: the law is not infallible but the recognition of a genetic dependence will help us greatly in the attempt to develop ore deposits. The critics, it should be noted, have not attempted to undertake any similar research. I hope that Dr. Newhouse may continue this line of investigation, and I am sure that this will reveal many guides to the mining geologist.

W. H. Newhouse (written discussion).—The examples mentioned in the discussion are most interesting. In connection with the asbestos deposits described by Mr. Wilson, it may be of interest to mention the not unusual occurrence of crocidolite asbestos on low anticlinal arches in South Africa.[112]

The contention that orebodies are found chiefly on tops of anticlines because they are more frequently exposed while synclines are hidden will account for but few of the occurrences. It is difficult to see how it can have any bearing at all on the smaller structures. With the larger structures such a theory calls for a special case of the relation of topography to structure, the common existence of which detailed examination of maps and cross-sections does not sustain. The reader can corroborate this statement by examining the maps of mining districts; all degrees of adjustment of topography to structure, as well as lack of adjustment and all degrees of development of the cycle of erosion, may be found. This might well be expected in view of the complex geological history of most ore-bearing districts.

[111] W. Lindgren: Research in Processes of Ore Deposition. *Trans.* A. I. M. E. (1928) **76**, 290–307.

[112] A. L. Du Toit: The Geology of South Africa, 393. Edinburgh, 1926. Oliver & Boyd.

Age and Structure of the Vein Systems at Butte, Montana

By James C. Ray, Stanford University, Calif.

(San Francisco Meeting, October, 1929)

The age classification of the mineralized veins of the Butte district, as given by Weed and Sales, was tacitly accepted for many years. Weed, whose field work was completed in 1906, divided the copper veins into groups of four ages.[1] In 1913, Sales modified Weed's classification and assigned all mineralized veins (including the silver-zinc veins of the border or "peripheral" zones) to three periods of fracturing and mineralization.[2]

Both writers based their classifications mainly on the relative ages of the observable faulting whereby the veins of a so-called "earlier" system are intersected and offset by those of a "later" system. Nevertheless they possess structural and mineralogical characteristics in common which indicate that they are of the same age. The writer has maintained since 1914[3] that these vein systems were formed during one general period of mineralization along a network of intersecting fractures of approximately simultaneous origin and that the existing offsets are largely due to postmineral faulting.

Leith,[4] takes cognizance of the accumulating mass of data relative to the simultaneous formation of intersecting fracture systems and says, referring to Butte: "It now appears that the mineralization accompanying them (individual structure systems into which the veins have been grouped) is substantially of one period, indicating that the successive structural movements took place mainly within the period of mineralization."

This paper presents the results of certain structural experiments which suggest a modified interpretation of the published data on the Butte district. The interpretation is substantiated by the writer's observations in the Butte mines and is offered with the hope that it will reconcile the structural with the mineralogical conditions existing in this most interesting district; also, that it will emphasize the importance of a

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[1] W. H. Weed: Geology and Ore Deposits of the Butte District, Montana. U. S. Geol. Survey *Prof. Paper* 74 (1912) 59.

[2] R. H. Sales: Ore Deposits at Butte, Mont. *Trans.* A. I. M. E. (1914) **46**, 12.

[3] J. C. Ray: Unpublished lecture, A. I. M. E. Meeting, Salt Lake City, August, 1914.

[4] C. K. Leith: Structural Geology. New York, 1923. Henry Holt and Co.

closer study and correlation of the relations between initial fracturing, mineralization, and faulting of ore deposits in general.

Development of Fracture and Fault Systems

The causes of fractures and faults have been the subject of investigation since early in the nineteenth century and the results have been ably summarized by Leith.[5] Unfortunately, many geologists whose work has been more particularly concerned with the economic branch have been prone to neglect the broader phases of geologic structure and their direct influence on the origin and more localized structural aspects of ore deposits.

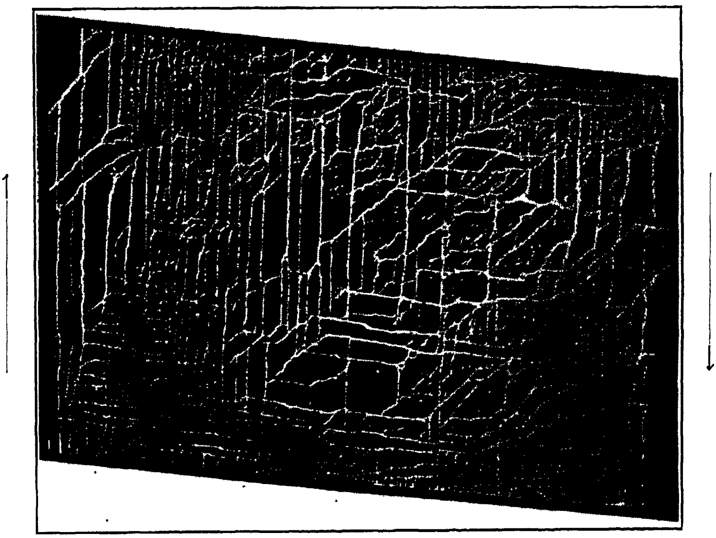

Fig. 1.—Fractures produced in paraffin coat on rubber sheet by shearing (Mead).

It is now generally recognized that homogeneous crystalline rock masses, when subjected to sufficient strain, are fractured and faulted in a systematic manner and that the local manifestations of the resulting forces are often the secondary or complimentary rather than the major ones. It may also be accepted that tension fractures will tend to remain open and that those resulting from compression will show the opposite tendency while the stresses which caused fracture remain active. When the stresses become dissipated the resulting blocks will tend to settle or seek gravitational adjustment. Rotary or torsional strain will develop various combinations of compression and tension phenomena.

[5] C. K. Leith: *Loc. cit.*, 356.

These general principles have been satisfactorily demonstrated by Becker[6] and Mead.[7]

Fig. 1 shows Mead's mechanical reproduction of the development of intersecting fractures due to shearing. He explains:[8] "The first fractures to appear in any one locality on the rubber sheet are usually tension cracks inclined about 45° to the direction of the shearing movement. These are at right angles to the direction of the maximum

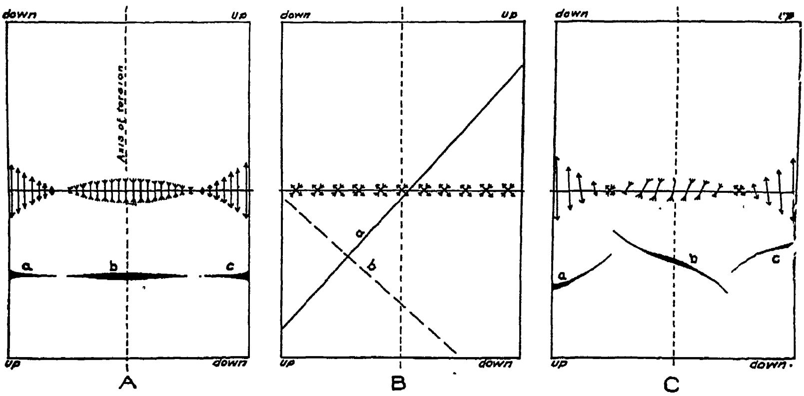

FIG. 2.—STRESSES AND FRACTURES CAUSED BY TORSIONAL WARPING (MEAD).

A. Stresses caused by local changes in area due to torsional warping, while total area has remained constant. Arrows indicate direction and relative magnitude of stresses along any transverse line. Heavy shaded lines indicate direction and relative magnitude of resulting tension cracks at *a* and *c* and of compression fractures at *b*.

B. Stresses developed on upper surface of a layer, due to bending caused by torsional warping. Arrows indicate direction and relative magnitude of stresses along any transverse line. Direction of tension cracks on upper surface is indicated by line *a*; on lower surface, by line *b*.

C. Resultant of stresses shown in *A* and *B*. Arrows indicate direction and relative magnitudes of resultant forces along any transverse line. Resulting type of tension cracks is shown by lines *a* and *c*. Line *b* indicates position and magnitude of compression phenomena.

elongation and appear as vertical open cracks. They are followed immediately by two sets of vertical faults with horizontal displacement, one set striking parallel to the direction of movement and the other parallel to the free edges of the rubber sheet. These represent two directions of non-distortion or two shear planes developed by the shearing movement in which direction of relief is in the plane of the paraffin layer."

Fig. 2 is Mead's diagrammatical representation of the stresses and resulting fractures due to torsional warping.

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[6] G. F. Becker: Finite Strain in Rocks. *Bull.* Geol. Soc. Amer., (1892) **4**, 50. Simultaneous Joints. *Proc.* Wash. Acad. Sci. (1905) **7**, 267.

[7] W. J. Mead: Mechanics of Geologic Structures. *Jnl.* Geol. (1920) **28**, 505.

[8] W. J. Mead: *Loc. cit.*, 512.

When diastrophism is active there can be little doubt that compression, tension, shearing and torsional warping are all brought into play and must necessarily modify the results.

Formation of Butte Fracture Systems

Application of the principles illustrated in Figs. 1 and 2 to a study of the map of the vein net at Butte, Fig. 3, leads to the supposition that the Butte fracture systems may have been formed in a similar manner. Add to this the hypothesis that hypogene ore deposition took place simultaneously in the three systems and the supposition must become a conclusion. Leith[9] in discussing Mead's work says: "Detailed study of the complex vein and fault system in the homogeneous Butte granite suggests strongly that these structures may be due to some sort of a progressive shearing movement of the kind above indicated."

The formation of fractures in the upper crystalline shell of an enormous intrusive magma, such as the Boulder batholith, can be ascribed to three general causes:

1. Cooling and shrinkage of the slowly crystallizing magma.
2. Local doming or settling of the upper crystalline shell due to drawing off of portions of the still viscous magma.
3. Broader phases of diastrophism which may or may not be complicated locally by conditions as outlined in 1 and 2.

Cooling or shrinkage jointing is very marked in the Butte granite. It is earlier than the vein systems and, from the point of genesis is unrelated to them,[10] although reopening along these earlier fractures near the later fissures has sometimes resulted in their mineralization.

Fracturing due to doming or settling of the upper crystalline shell formed passageways for the aplite and quartz porphyry which are squeezed into the quartz monzonite (Butte granite) of the district in the form of dikes. These dikes are earlier than the veins and are evidently phases of magmatic differentiation which took place before concentration of the mineralizers and at a higher horizon in the viscous magma. Increasingly deep development of the mines discloses that in some instances the quartz-porphyry dikes widen out to massive proportions and that the quartz porphyry grades into the normal quartz monzonite, suggesting magmatic differentiation *in situ*. Sagging of the crystalline shell and settling of the blocks into the underlying molten magma seems to be indicated by the widening of the dikes with depth. If this is the case, the fractures were due to tension.

Consideration of the broader features of the physiographic history of the region leads to the conclusion that diastrophism was in progress when the

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[9] C. K. Leith: *Loc. cit.*, 45.
[10] R. H. Sales: *Loc. cit.*, 8.

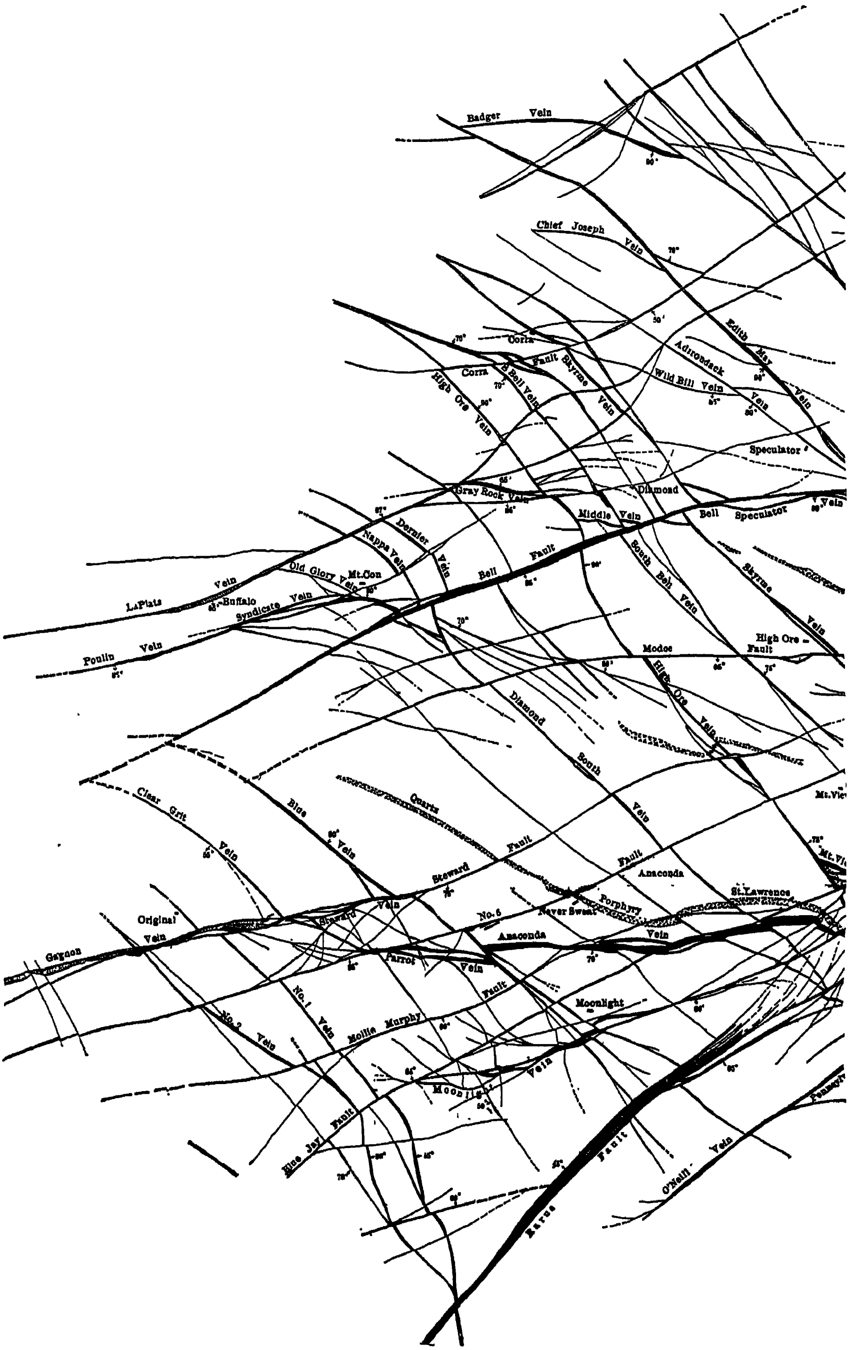
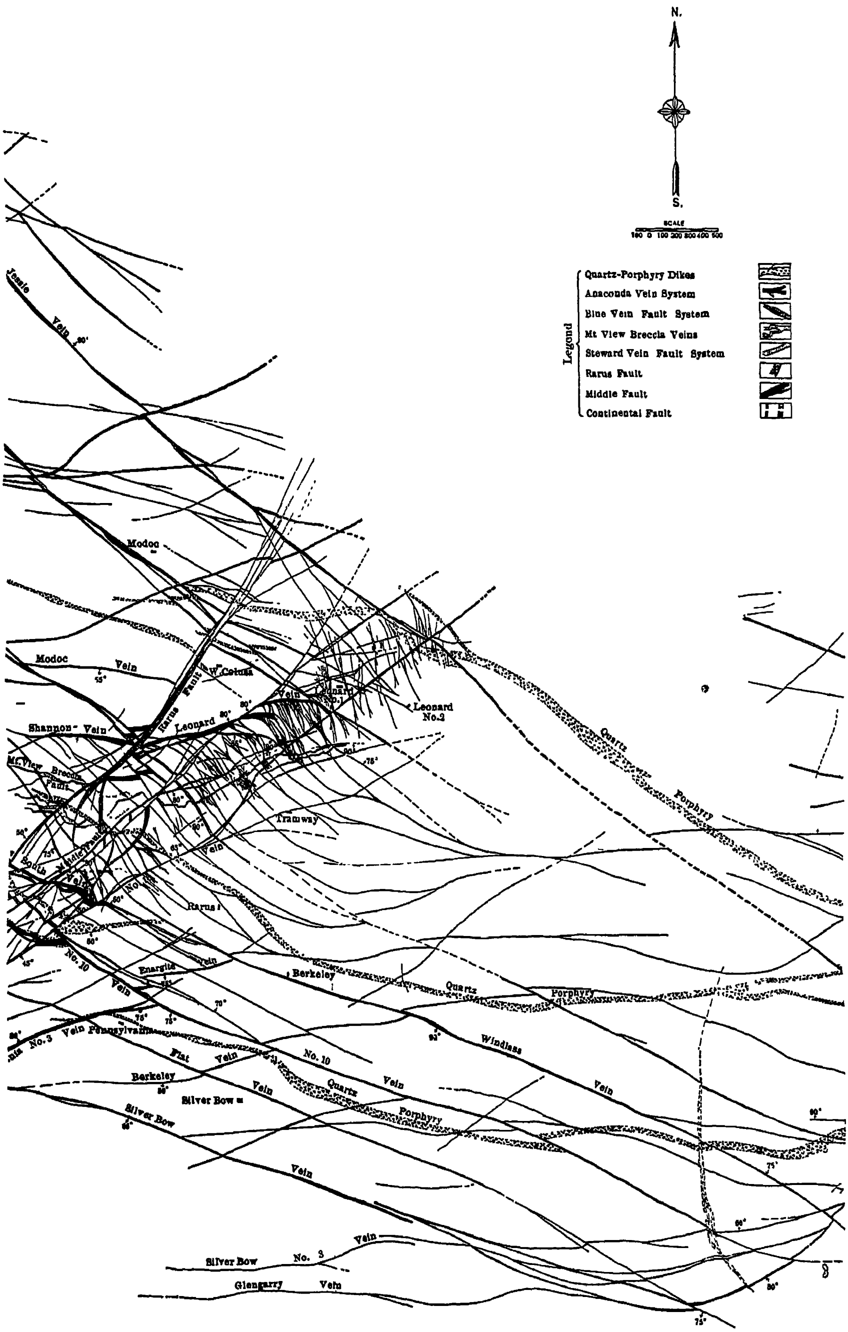


FIG. 3.—HORIZONTAL SECTION OF BUTTE DISTRICT (4600 FT. ABOVE SEA LEVEL, 1500



FT BELOW SURFACE) SHOWING STRUCTURAL RELATIONS OF FISSURE SYSTEMS (SALES).

Butte fracture systems were developed and that they were a local phase of the broader activity. The richness of the Butte deposits may be attributed to the fact that these local structural features were profound and that chance so placed them that they tapped a reservoir of unusually high copper concentration. Billingsley[11] points out that the early hypogene mineralization at Butte is similar to that at numerous other localities in the Boulder batholith but at Butte alone copper concentration by magmatic solutions was abnormal.

For the purposes of this paper the principal items of geologic interest are the intrusion of the Boulder batholith and the earth movements of the region in general. These, as compiled from papers by Billingsley[12] and Atwood,[13] may be summarized as follows:

1. Late or Upper Cretaceous.—Mountain growth; development of sharp folds; thrust faulting along northwest lines.
2. Late or Upper Cretaceous.—Andesitic flows which in part became the roof of the Boulder batholith.
3. Late Cretaceous or Eocene (?).—*Intrusion of the Boulder batholith.*
4. Oligocene.—Uplift of Rocky Mountain province; development of grea intermontane troughs; normal north-south faulting.
5. Miocene-Early Pliocene.—Rhyolite and dacite flows; uplift of mountain masses.
6. Pleistocene.—Faulting (Continental fault and Rarus fault?).

It is thus seen that the area now occupied by the Butte district is part of a region that has been subjected to diastrophism at several periods since the intrusion of the Boulder batholith. The batholith itself was intruded during the late Cretaceous or early Eocene period, but the Butte ore deposits could hardly have been formed until some time later. Quartz-porphyry dikes were intruded into the already crystallized shell of the magmatic intrusion. With depth these dikes give way to magmatic differentiation *in situ* (quartz porphyry) which, with continued crystallization, becomes part of the solidified crust. Shrinkage jointing was developed in the slowly cooling rock. Crystallization and cooling become retarded by insulation caused by the increasing thickness of the shell. Thus the magmatic reservoir of the concentrated mineralizers is driven ever deeper. In the appalling magnitude of these natural phenomena, magmatic differentiation, crystallization and the concentration of mineralizers, time must be estimated in geologic periods rather than in terms of thousands of years.

After intrusion of the Boulder batholith, the next profound adjustment took place during the Oligocene period, when the region was subjected to uplift and the formation of the great intermontane troughs accompanied

[11] P. Billingsley: The Boulder Batholith of Montana. *Trans.* A. I. M. E. (1915) **51,** 46.

[12] P. Billingsley: *Loc. cit.*, 35.

[13] W. W. Atwood: Physiographic Conditions at Butte, Mont., and Bingham Canyon, Utah, when the Copper Ores in These Districts were Enriched. *Econ. Geol.* (1916) **11,** 697.

by north-south faulting on a large scale. Here then we find diastrophism on a scale sufficient to develop internal fracture systems within the north-south fault blocks. There can be little doubt that fracture systems of the Butte type can be produced by strains brought into play during the adjustment of regional fault blocks. The writer ascribes the Butte mineralized fractures to such a cause and places both the formation of the fractures and their hypogene mineralization in the Oligocene period.

From late Cretaceous or early Eocene to the Oligocene may surely be considered sufficient geologic time to have accomplished magmatic differentiation, crystallization of a granitic shell probably more than 20,000 ft. thick, and the concentration of a vast store of mineralizers.

Regarding the depth to which these fractures could have remained open, Lindgren[14] gives credence to Adams' deductions[15] that: "Cavities may exist in granite to a depth of 11 miles," and supplements this with Bridgman's[16] calculations to show that it is "extremely probable that minute crevices, at least large enough for the percolation of liquids, exist in the stronger rocks at depths corresponding to pressures of 6000 to 7000 kg. per sq. cm." If this be true it would be entirely possible for the Butte fractures to remain open at depths of 4 or 5 miles.

Ore Deposition at Butte

The theory is now well established that ore deposits of the vein-replacement and fissure-filling types, when they occur within or in close proximity to extensive batholithic intrusions, have received their mineral contributions from the intrusive magma. The mineralizers must have escaped from their reservoir of accumulation and traveled to the zone of deposition along fissures which were of necessity continuous open channels of circulation. The east-west (Anaconda system) veins at Butte supply these necessary channels.

The massive crystalline texture and the vuggy structure in the east-west veins indicate that much of this ore was deposited between widely separated walls and sometimes as interstitial filling between fragments of country rock which had settled between the walls of the fractures. This actual separation of the walls over long distances is indicative of tensional stresses. In the eastern portion of the district the east-west veins lose their identity in a maze of branching fractures, the result of shear. The action of this shear would certainly pull apart the walls of the fractures occurring to the west. (See Fig. 3.)

[14] W. Lindgren: Mineral Deposits, 39 and 42. New York, 1928. McGraw-Hill Book Co.

[15] F. D. Adams: An Experimental Contribution to the Question of the Depth of the Zone of Flow in the Earth's Crust. *Jnl. Geol.* (1912) **20**, 97.

[16] P. W. Bridgman: Failure of Cavities in Crystals and Rocks under Pressure. *Amer. Jnl. Sci.* [4] (1918) **45**, 243.

The characteristic relations of the Butte veins warrant the assumption that the east-west veins were the first and principal fracture zones along which opening and initial movement took place as a result of tension; and that the northwest (Blue) and northeast (Steward) systems were noncontinuous secondary fractures developed by compression within the local fault blocks; and further, that relatively little movement took place along these secondary fractures until after hypogene mineralization was completed. Considerable posthypogene faulting is evidenced by clay gouge and freshly slickensided walls in the secondary fractures. It is also possible that faulting took place simultaneously along both systems of secondary fracturing. This later faulting could have occurred during both the Early Pliocene and Pleistocene periods.

Deductions from Experimental Model

The model illustrated (Figs. 4 and 5) was constructed to investigate the hypothesis that movement could have taken place simultaneously along the two secondary fracture systems. A rectangular block of wood

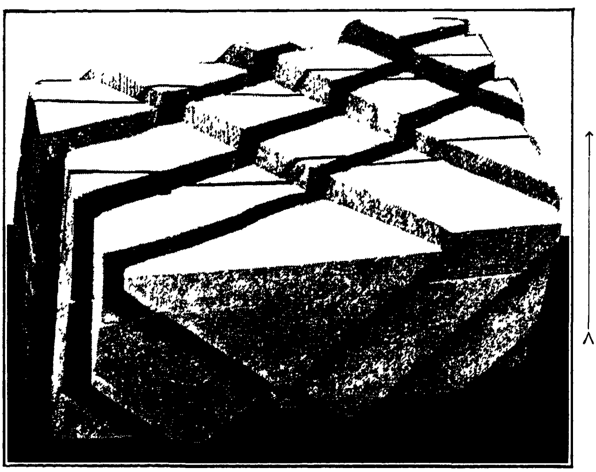


Fig. 4.—Simultaneous settling of sawed blocks along two sets of intersecting planes.

was sawed lengthwise and the three resulting sections were glued together. These glued saw cuts were allowed to represent the east-west (Anaconda) vein system. The assembled block was again cut in two directions to represent the northwest and northeast systems, using the approximate strikes and dips of the actual veins. The resulting disjointed segments were now held together in their normal positions with rubber bands and manipulated to simulate the actual recorded relations of the Butte veins. Cross-sections and data given by Weed and Sales show that the

faulting in the northwest and northeast veins was normal; *i. e.*, the hanging walls moved downward in relation to the foot walls. Sales provisionally placed the Bell fault in the Steward system[17] but the fact that it is a reverse fault and is not known to contain indigenous ore would seem to exclude it from this system, in which the veins otherwise show a systematic and orderly duplication of normal faulting.

By trial with the sawed segments, it was found that resistance applied horizontally from the northwest maintained the continuity of the northeast planes while the blocks were induced to settle simultaneously along both the northeast and northwest systems. The northwest "veins," on the other hand, suffered displacement where intersected by

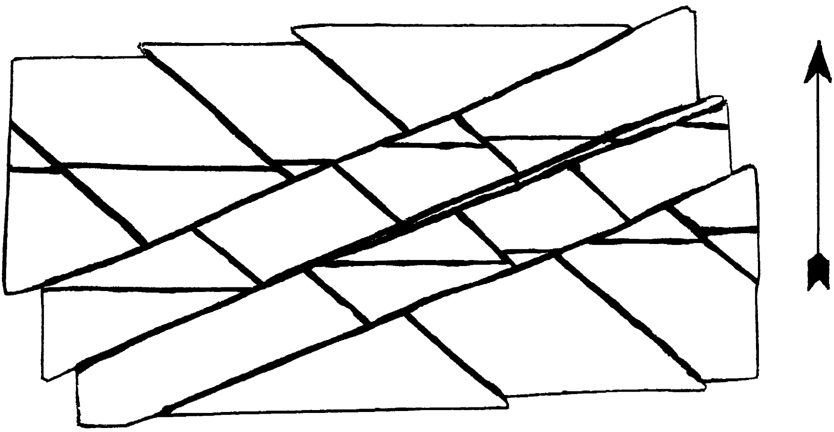


FIG. 5.—HORIZONTAL PLAN OF "VEIN" NET DEVELOPED BY SIMULTANEOUS SETTLING OF BLOCKS ILLUSTRATED IN FIG. 4.

the northeasters. The east-west "veins" were offset at their intersections with the other two systems.

The complicated block system was now glued together and again cut horizontally. Fig. 4 shows the adjusted blocks with the several fault planes and their intersections with the various exposed intrablock surfaces. Fig. 5 is a photograph of the vein net as exposed by the horizontal cut. This pattern reproduces all the characteristic offsets of the actual mineralized veins in the Butte district, excepting, of course, the shear phenomena of the horsetail structure described by Sales.[18]

Comparison of Fig. 5 with Fig. 3, which is a reproduction of Sales' horizontal plan showing the structural relations of the Butte veins 1500 ft. below the surface,[19] discloses:

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[17] R. H. Sales: *Loc. cit.*, 22.
[18] R. H. Sales: *Loc. cit.*, 17.
[19] *Loc. cit.*, Plate I.

1. The same shifting to the northwest of each successive segment of the east-west veins where intersected by the northwesters;

2. The same offset to the eastward of the progressively southern segments of the northwest veins where intersected by the northeasters, and

3. The same separation of contiguous east-west segments when intersected by the northeasters.

It is quite true that structurally the relations as shown on Sales' map (Fig. 3) could have been produced by three distinct periods of vein formation, the veins of a later period intersecting and faulting those of earlier periods. If this interpretation be accepted, certain corollary conditions should be expected:

1. The solutions that mineralized each later set of fractures must have left some evidence of superimposed mineralization on the filling of the earlier veins, at least at or near their intersections;

2. Hypogene mineralization having been completed in the earlier veins, the later veins could hardly be expected to exhibit the same sequence of minerals, which in the earlier veins is admittedly the result of changing conditions that have produced progressively quartz, pyrite, sphalerite, tetrahedrite and enargite (these being only the more important hypogene minerals);

3. If the veins of the later systems were formed in fault fissures, it is to be expected, even though the orebodies themselves be of comparatively limited extent, that the barren portions of these veins would at least show some deposition of gangue minerals, such as vein quartz and pyrite, along their courses.

None of these expected conditions exist. Deposition of a second or third sequence of hypogene minerals is conspicuously absent. The writer has searched fruitlessly on numerous occasions for this evidence. At and near intersections of mineralized northwesters with east-west veins there is no evidence of a second generation of quartz, pyrite, sphalerite, tetrahedrite or enargite, yet this same sequence is present in both systems of veins. The same is true of the mineral relations between east-west and northeast veins.

Further, the hypogene mineral sequence is the same in all veins for any given part of the district. Sales says:[20] "The mineral composition of the ores of the various vein systems is of marked similarity. A suite of hand specimens typifying the ores of the Anaconda system does not differ materially from a similar suite collected from the later veins." He also writes,[21] " . . . and again a feature of interest is found in the fact that within the area there is no essential difference in the minera-

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[20] R. H. Sales: *Loc. cit.*, 62.
[21] R. H. Sales: *Loc. cit.*, 59.

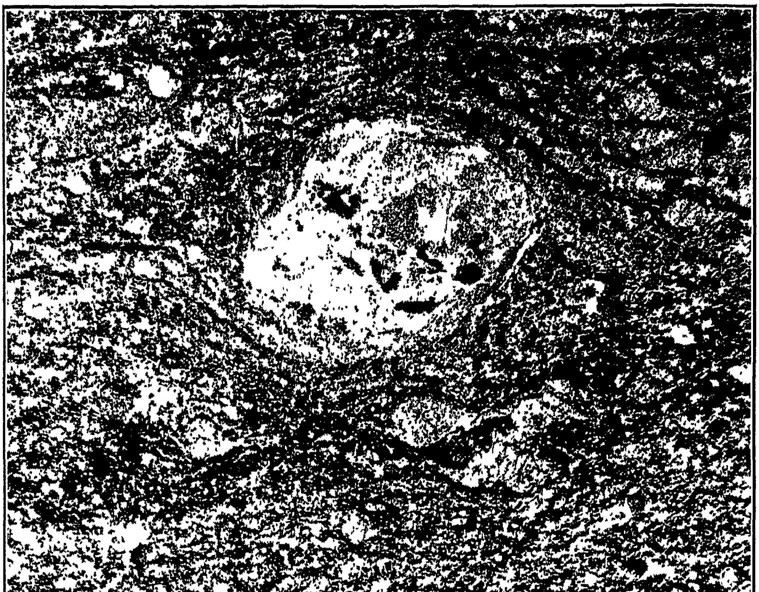
logical composition of the vein filling in veins of the different ages . . . In one locality a vein of the Anaconda system may contain proportionately more enargite than a near-by vein of the Blue system, while in other localities the reverse is true." This conclusion the writer has verified by extensive microscopic study of polished sections of the Butte ores.

Referring to the vein filling of the northwest and northeast vein systems, Sales writes:[22] "The ore of the Blue and Steward fissures occurs in the form of 'shoots' which vary greatly in size and extent. These oreshoots are irregular in outline and are separated on the strike of the fissure by hundreds or even thousands of feet of barren crushed granite and fault clay composing the fissure zone. The walls of the ore are seldom free from fault gouge or other evidences of extensive movement." Due to these conditions, the author has long held the opinion that the Blue (northwest) and Steward (northeast) vein systems were not formed initially as continuous fault fissures but that the ores were deposited only in certain areas along zones of maximum compression where intra-fault-block adjustment or slight local movement relieved the pressure sufficiently to form open channels of limited extent for circulation of the mineralizers. In other places actual fractures were absent along the courses of the compression zones.

The amount of clay gouge and the condition of slickensided walls in the secondary systems indicate postmineral movement of considerable intensity. Contrary to the generally accepted idea, this gouge does not show evidence of replacement by ore minerals. The material supposed to be ore replacement of clay gouge is in fact a black gouge composed of finely comminuted particles of hypogene ore minerals, quartz, pyrite, enargite, showing only incipient replacement by undoubted supergene (downward secondary) chalcocite (Fig. 6). This gouge was 2 to 3 in. thick, was contiguous to a slickensided surface of solid ore which contained the same minerals, and was of the same thickness as light colored clay gouge that occurred farther along the wall of the vein. The specimen was collected from the 1600-ft. level of the Edith May (northwest) vein in the Badger mine. From these data, it seems logical to conclude that the movement recorded in this vein is postmineral in age.

Weed and Sales are very clear in their statements that the several sets or systems of veins are distinct structural units which were developed independently. Sales, however, infers a self-contradiction when he says:[23] "Structurally there is no good evidence for distinct periods of mineralization in the Butte veins. It is here held that there was but one period

[22] R. H. Sales: *Loc. cit.*, 78.
[23] R. H. Sales: *Loc. cit.*, 62.



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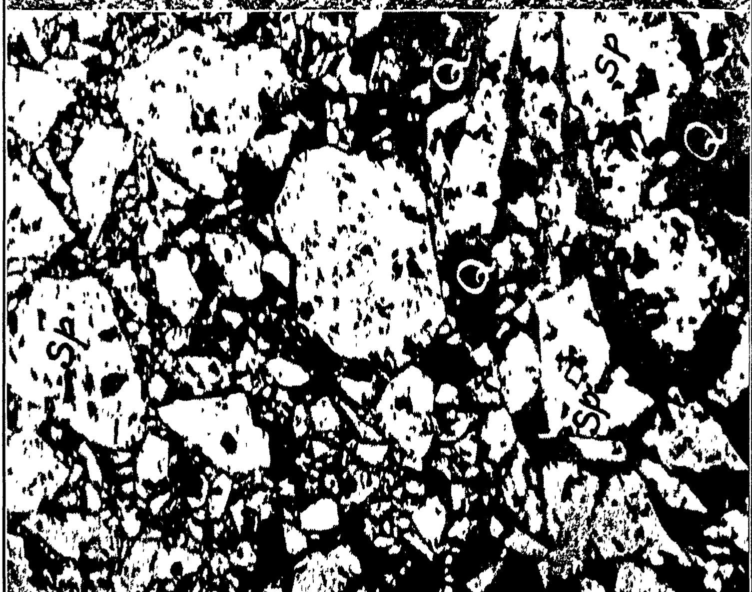

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FIG. 6.—GOUGE FROM EDITH MAY VEIN, BUTTE. $\times$ 30.
Composed of enargite, pyrite and a small amount of quartz cemented with supergene chalcocite. From 1600-ft. level, Badger mine.

FIG. 7.—FINELY COMMINUTED SPHALERITE CEMENTED WITH HYPOGENE QUARTZ. $\times$ 150.
From 1000-ft. level, Rainbow Lode, Black Rock mine, Butte.

of mineralization, varying in intensity, possibly, from time to time, with important changes in chemical character of solutions." This can hardly be reconciled with his statement referring to Group B, under which he classifies the northwest and northeast systems as "persistent well-defined fissures of marked displacement and of later age than Group A" (Anaconda or east-west system).[24]

It seems hardly possible that one force causing fractures and movement in a northwesterly direction could become dissipated and that a second force could later become active and cause fractures and movement in a northeasterly direction and in the same area, all within a single period of hypogene mineralization. On the other hand, we have seen that there were three widely separated geologic periods during which diastrophism was active.

Leith, in more recent years, recognizes that fissuring and mineralization of the three vein systems are closely related in regard to the time element. He writes:[25] "In the past these sets of faults have been treated as separate structural units developing separately. It now appears that the mineralization accompanying them is substantially of one period, indicating that the successive structural movements took place mainly within the period of mineralization. It further appears that the clean cut separation into structural units fails to take into account the existence of many branches, cross-overs, and gradational features tending to tie the different systems together . . . The first two are probably somewhat complementary, and related to a single major and progressive shear of the district . . . "

The writer has demonstrated that the second and third systems (northwest and northeast) could both be complementary to the first (east-west) system. It has been demonstrated with the experimental blocks (Figs. 4 and 5) that the vein net could have been formed in this way and that the displacement by the second and third sets could have taken place principally after hypogene mineralization. Simultaneous fracturing in several directions has long been recognized by Becker[26] who cautions: "It is often assumed that when one fissure faults another the latter is the older, but this inference is not justifiable and they must often be of exactly the same age." He further states his doubt: "whether in a region once jointed by a system of forces, the application of a new system of forces could produce a fresh set of joints systematically arranged."

Weed seems to have had this idea in mind when he said, regarding Butte,[27] "From the intersection of so many fissures it is evident that the

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[24] R. H. Sales: *Loc. cit.*, 14.
[25] C. K. Leith: *Loc. cit.*, 88.
[26] G. F. Becker; *Loc. cit.*
[27] W. H. Weed: *Loc. cit.*, 63.

whole district is netted and that there has possibly been a mutual displacement by intersecting fissures of the same age . . . ;" but he argues himself out of this conclusion by his failure to recognize all hypogene mineralization as belonging to one period, that the displacements by the two secondary sets of fissuring could actually have been simultaneous, and that the observed faulting could be of postmineral age.

Movement during Hypogene Mineralization

Evidence does not support the conclusion that there was no movement whatsoever in the fissures during the period of hypogene mineralization. That there was such movement is shown by many examples of brecciated ore fragments and ore minerals which have been cemented by later hypogene minerals. Fig. 7 shows quartz, a hypogene mineral, cementing finely comminuted sphalerite. This ore is now a hard flinty material but before cementation by the quartz it was undoubtedly more or less gougelike in composition. Sphalerite has well-defined cleavage planes and only slight movement would serve to grind it into small fragments like those in Fig. 7.

In general, cemented breccias of the Butte ores have quartz or other undoubted hypogene minerals as the matrix. Gouge, as such, is uncemented except where it is composed largely of hypogene sulfides and it is then sometimes partially consolidated by replacement of the finer particles by supergene chalcocite. It may be assumed therefore that gouge formed during the period of hypogene mineralization became cemented or replaced by minerals of that period and that all gouge now occurring as such is the result of posthypogene movement.

There is no doubt that the period of mountain uplift which occurred during the early Pliocene could account for much of the posthypogene faulting in the Butte area. The Pleistocene furnishes evidence of still another period of extensive faulting.

If we consider that the east-west fractures were formed by tension and that they were the main channels for circulation and deposition of hypogene minerals, we must believe that they were strongly healed before secession of the early mineralizers, even though there may have been considerable movement during this period. On the other hand, the complementary systems (northwest and northeast), being the result of compression, offered only indifferent channels for circulation at points of localized intrablock adjustment. Along these zones of maximum compression, planes of weakness were created which yielded easily to later forces to form true fault planes. It is the movement along these later developed fracture planes that is illustrated by the writer's model. Movement could have been simultaneous along the two sets of planes.

Conclusions

The following conclusions are supported by the actual conditions in the Butte district. They reconcile and coordinate physiographic, structural and mineralogical data which, when considered independently, have sometimes appeared to offer contradictory evidence.

1. The three mineralized vein systems of the Butte district are of the same age and were formed during a single period of structural readjustment.
2. The observable faulting along the northwest and northeast vein systems could have taken place simultaneously.
3. The northwest and northeast vein systems were not continuous fractures at the time of their mineralization but were compression zones along which actual fractures existed only where minor intrafault-block adjustments caused comparatively slight movements.
4. The east-west (Anaconda) veins were formed in tension fractures which were the principal channels of circulation and from which were tapped such mineralizers as found their way into and formed the ores of the northwest and northeast veins.
5. Forces active during posthypogene mineralization formed the present continuity of the northwest and northeast veins. This continuity did not exist at the time of their hypogene mineralization or the existing barren brecciated zones along their present courses must have been mineralized.
6. Hypogene mineralization occurred as a result of and in conjunction with regional diastrophism during the Oligocene period.
7. Regional diastrophism that occurred during the Pliocene and Pleistocene periods can well account for the posthypogene reopening and faulting of the mineralized vein systems as well as the formation of the later unmineralized faults of the Butte district.

Postulations

A fracture or fissure is not a vein until such fracture or fissure has become mineralized.

The age of a vein should be determined by the age of its mineralization and not by the age of the fracture in which the mineralization takes place.

Intersecting fractures may be of different ages but if they all are mineralized at the same time the resulting veins are of the same age.

Geology of the Parral Area of the Parral District, Chihuahua, Mexico

By Harrison Schmitt,* Hanover, N. M.

(New York Meeting, February, 1930)

The Parral area, a part of the Parral mining district of Southern Chihuahua, is situated in and near the City of Parral, with the most important mine of the district, La Prieta, lying within the city limits. The famous Veta Colorado and Palmilla veins near Villa Escobedo or Minas Nuevas are a few kilometers to the northeast (see Fig. 1). The chief metals produced are silver, lead and zinc, with some copper and gold.

This area was one of the earliest to produce metal in Mexico, according to records at Parral, which show that mining started in 1632. In near-by Santa Barbara, however, mining had begun 85 years earlier. It is said that when silver was discovered in the Veta Colorado, the Santa Barbara gold deposits, at that time nearly exhausted, were immediately abandoned for the more profitable silver mines at Minas Nuevas (New Mines).

During the post-revolutionary period, 1923–1928, the author made many trips to the Parral district to do mine examination work for several units of the American Smelting and Refining Co. This paper is an abstract of some of the geologic data collected, together with the results of a microscopic study of the ores and rocks—the latter investigation made at the University of Minnesota—and is intended to supplement one recently published.[1]

Topography

The Parral district lies in the eastern foothills belt of the Sierra Madre Oeste. The area is about evenly divided between fairly flat to rolling plains and mesas on the south and rounded hills to rough precipitous slopes on the west and north. The relief is approximately 550 m. (1800 ft.).

The areas of intrusive monzonite and Jurassic (?) sedimentary rocks give rise to a topography of low rounded hills and relatively flat valleys; as contrasted with the volcanics, with their alternating beds of soft

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[1] H. Schmitt: Geologic Notes on the Santa Barbara Area of the Parral District. *Eng. & Min. Jnl.* (1928) **126**, 410.

tuffs and hard flows, which have yielded extremely sharp erosional forms that usually express the underlying fault-block structure (Fig. 2).

Broad mesas occur here and there. One of these, just southeast of Parral, has a width of more than 1 km. (0.6 mile). These are remnants of

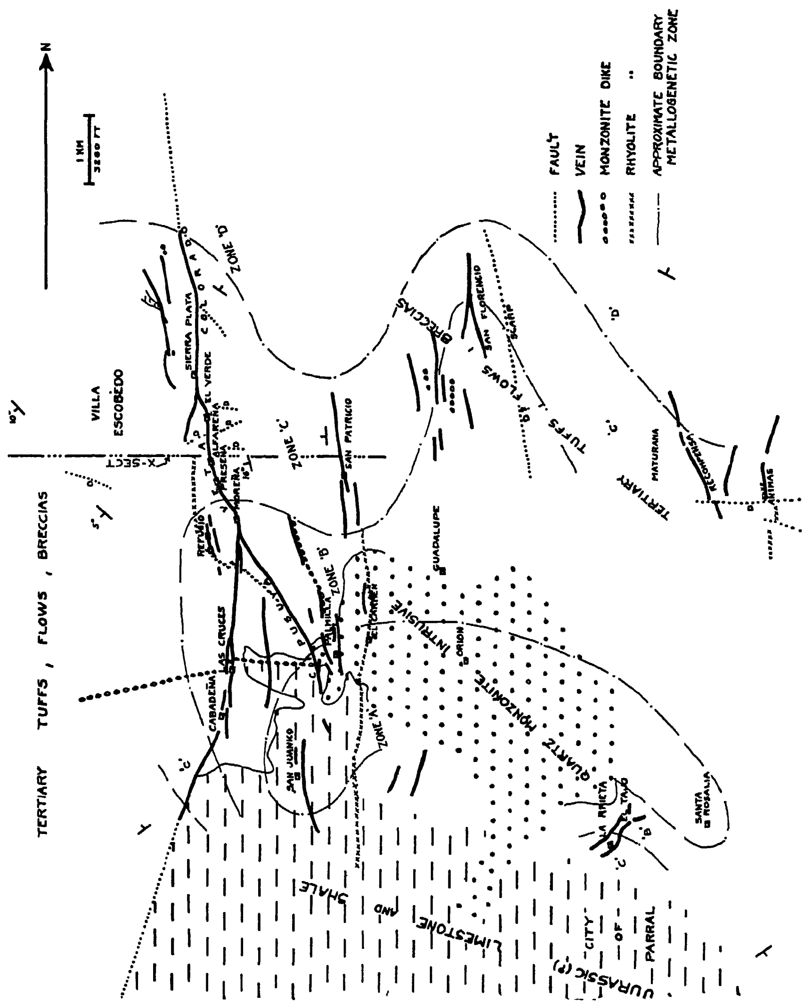

FIG. 1.—SURFACE GEOLOGY, PARRAL AREA, SHOWING METALLOGENETIC ZONES.

the old Santa Barbara erosion surface that once occupied the area but is now largely destroyed.

GEOLOGY

The earliest geologic event represented by the exposed rocks of the district was the deposition of a series of limestones and shales, which probably took place in Jurassic time. Later these were folded, uplifted

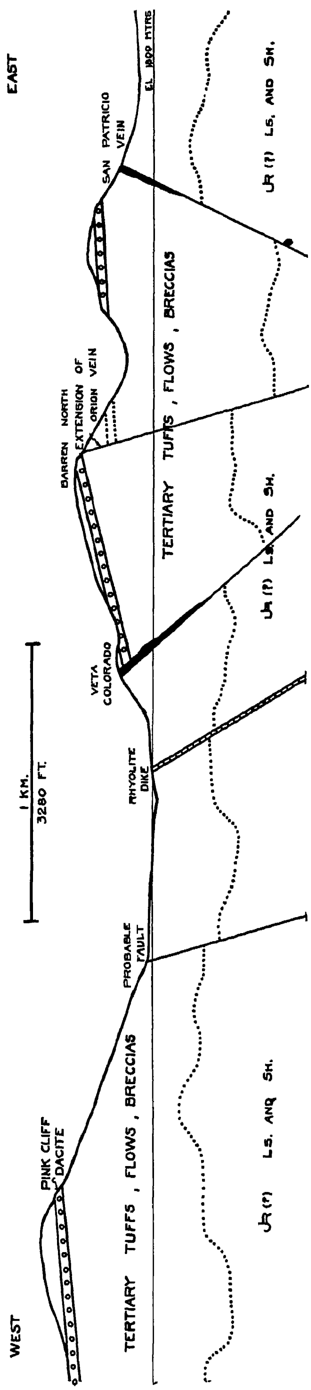

FIG. 2.—EAST-WEST SECTION THROUGH VETA COLORADO AND NEIGHBORING COUNTRY. LOCATION OF SECTION IS SHOWN BY FIG. 1.

and deeply eroded into a mountainous topography. This topography was buried, probably in Tertiary time, by a thick series of volcanic rocks consistng of breccias, flows and tuffs.

The volcanic and sedimentary rocks are broken by great normal faults that generally strike north-south and have dip slips as great as 450 m. (1500 ft.). These faults cut up the country into elongate north-south blocks, the ultimate result being the depression of the center of the area, possibly because the continued extravasation removed large quantities of volcanic material from directly below. Erosion has developed most of the present surface on these rocks.

The Parral area appears to have been a center of the regional volcanic activity, having one main vent and several smaller satellitic ones. This idea is supported by the petrographic consanguinity of the main Parral stock with the neighboring volcanic rocks, and also by the thinning out and final disappearance of the volcanics on the average of about 16 km. (10 miles) away from the central stock. The volcanic center idea is further supported by the quaquaversal dip of the volcanic rocks away from the central stock, which very likely is depositional dip.

THE ROCKS

Sediments

The Jurassic (?) sediments are similar in character to those at Santa Barbara and are composed of blue-black thin-bedded limestones and limy shales, all apparently nonfossiliferous.

These sedimentary rocks are provisionally dated Jurassic because rocks

of similar character in southwestern Chihuahua have been called Jurassic, and because at Boquilla dam, about 80 km. (50 miles) north of Parral, they are overlain unconformably by massive limestones of supposed Comanchean age that have been less disturbed dynamically. No Paleozoic rocks are known in this part of Mexico, but it is probable that the Jurassic sea covered the entire area, because Jurassic sediments are known in near-by areas. All this indirect evidence points to the Jurassic age of the sediment at any rate, which, provisionally, may be called the Santa Barbara series, because of their excellent exposure at Santa Barbara.

Volcanic Rocks

The eruptive rocks are here called the Escobedo volcanic series because they are well exposed in the mountains west of Villa Escobedo. Their distribution in the Parral area is shown by Fig. 1.

The average composition of these rocks is andesitic, although they range from basalts to dacites.

A columnar section of the volcanic rocks measured about 1.6 km. (1 mile) west of Villa Escobedo is shown by Fig. 3. In general, the more basic rocks are found in the lower part of the series, the more acid near the top, there being a progressive increase in acidity from bottom to top of the series. At the very base is coarse breccia-conglomerate with limestone and quartzite boulders predominating, some of the boulders having diameters of 2 ft. or more. Just above this is much water-laid tuff and sandstone. Near the top of the series in the Villa Escobedo area (Fig. 3, No. 1) is a persistent flow about 35 m. (115 ft.) thick, which here is called the Pink Cliff dacite. This is an excellent marker because of its resistance to erosion and consequent good exposures, because of the characteristic green and black glass flows that are present just above and below it, and because it extends over a large area (Figs. 2 and 3).

Intrusives

Monzonite.—The most important intrusive of the district is the central monzonite stock (?) near Parral, the main mass of which is classed as biotite-quartz-monzonite porphyry. There are border phases more basic than this; F. W. Smith, in a private report, describes a younger granite phase in the Palmilla mine near the bottom of the main orebody. Macroscopically the monzonite is light gray in color. It contains prominent feldspar phenocrysts, of which those of plagioclase are the largest, sometimes being 3/8 in. long. Ferromagnesian minerals are relatively unimportant. Microscopically the rock is porphyritic in texture; the most abundant phenocrysts are zoned andesine and orthoclase, with minor biotite and augite. A few large apatite crystals are characteristic. The ground mass is usually xenomorphic quartz and orthoclase, but in

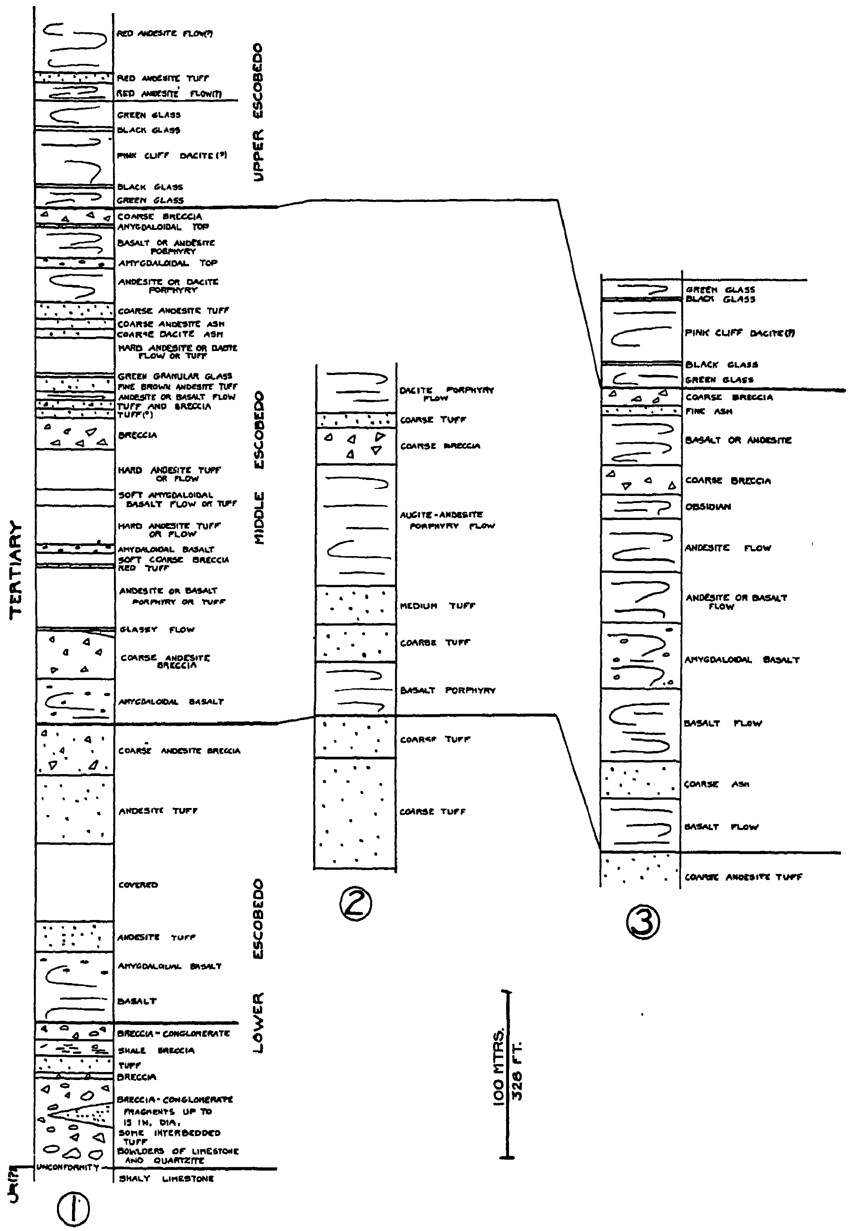

FIG. 3.—COLUMNAR SECTIONS IN PARRAL AREA MEASURED BY PLANE-TABLE SURVEYS.

1. Measured from Cabadeña shaft to the top of the mountain, which is 2 km. west of Villa Escobedo.
2. Measured in footwall shaft of Plata Verde mine.
3. Measured in hanging wall of Veta Colorado about ½ km. south of Alfareña shaft.

one specimen these minerals were graphically intergrown. Magnetite occurs in small amounts. In some cases the augite is altering to biotite; and locally, near mineralization foci, the alteration products sericite, chlorite, carbonates, kaolin and hematite have developed, and pyrite, specularite, carbonate and quartz have been introduced.

Rhyolite—North-south trending rhyolite dikes are fairly common in the Veta Colorado area. Some are earlier than the monzonite, but most are later. The very late ones fed surface flows of which remnants remain as caps of small mesas. Macroscopically the rhyolites are dense, pink, fine-grained rocks. One of the late dikes, near the El Carmen mine, shows conspicuous flow structure parallel to the contacts. Small elongated cavities parallel to the walls are filled with small crystals of quartz. One of the early dikes, cutting the Sierra Madre vein, shows $\pm$ 25 per cent. of quartz.

Basalt.—Fissure eruptions of basalt occurred in recent time. Intrusions of this material took the form of very thin irregular dikes and sills. A good section of this type of intrusion is exposed in a railway cut on the Parral and Durango R. R. near the Parral station. The host rock is the Jurassic (?).

Geologic History

Interpreting the columnar section in terms of geologic history, it appears that the Jurassic (?) sediments were folded, uplifted and eroded into mountains of sharp relief. This surface seems to have been inundated by a flood of volcanic rocks. The lava blocked the drainage and gave rise to closed basins that were rapidly filled with water-laid tuff and breccia-conglomerates, which, in turn, were covered by alternating flows and tuffs.

In Tertiary time, after the vulcanism had ceased, the volcanic rocks were deeply eroded. The drainage possibly developed on a volcanic plateau and the erosion seems to have been accompanied by uplift, because many of the streams are of the antecedent type, and cut across rocks and structure regardless of the paths of least resistance.

The very latest event, as noted above, probably occurring in historic time, was the fissure eruption of basalt that covered some of the recent topography with thin flows. This eruption seems to have been more important toward the southwest, especially near Santa Barbara.

This history appears common to southwestern Chihuahua and northern Durango; thus at Santa Eulalia, Comanchean limestone "mountains" can be seen buried by Tertiary volcanic rock. This indicates that the volcanic epoch was post-Comanchean. Again, at the Termopilas mine, about 32 km. (20 miles) southeast of Rosario, Durango, Jurassic (?) limestone "mountains" were observed, buried by the younger volcanic rocks.

Differentiation of Source Magma

Differentiation of the source magma that gave rise to the vulcanism of the district is suggested by:

1. Early basic and late acid phases in the central stock.
2. The following phenomena in the monzonite:

a. Zonal feldspars.

b. Micropegmatite.

c. Augite altered to biotite.

(*a, b* and *c* suggest that an increase of silica, alkalies and water occurred in the later stages.)

3. The progressive increase in acidity of the volcanic rocks from oldest to youngest, of which consanguinity with the monzonite intrusion is shown by:

a. Similarity of certain minerals in each; namely, large andesine, augite, and apatite crystals.

b. Similar average composition.

Structure

The regional structure is that of a broad, flat dome with the Parral stock (?) in the center. This dome has been block-faulted with the central part depressed, a graben, relative to the periphery. The block faults are normal and of great offset; 450 m. (1450 ft.) of dip slip was measured on one fault (Fig. 2).

In connection with the normal faults, especially along the Veta Colorado, it was noticed that three systems of fracturing were prominent near the vein-fault (Fig. 4). These are supposed to be the sets of fractures that often develop during normal faulting.[2]

The main faults, in most cases, have localized the mineralization in the Parral area. The Palmilla orebody, however, appears to have been controlled by hanging-wall tension fractures (Fig. 5).

The type of wall rock, apparently, controls the abundance, attitudes and susceptibility to mineralization of the hanging-wall and footwall fractures mentioned. With andesite walls the mineralization is closely confined to the main fault plane. With shaly limestone walls, however, there is irregular invasion of the walls by the mineralization. The cross-section of the San Juanico (Fig. 6) exemplifies this. In a similar manner, at Santa Barbara, where the walls of the veins are the same shaly limestone, many small hanging-wall and footwall veins are present.

The Palmilla oreshoot is in the monzonite hanging-wall of the Capusaya vein, which is the southern extension of the Veta Colorado vein-fault

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[2] B. Willis: Geologic Structures, Chap. VII. New York, 1923. McGraw-Hill Book Co.

(Fig. 1). This mineralization is different in form from that in the andesite or shaly limestone in this area. Indeed, in shape, relation to structure, and type, it resembles that of the Comstock lode.

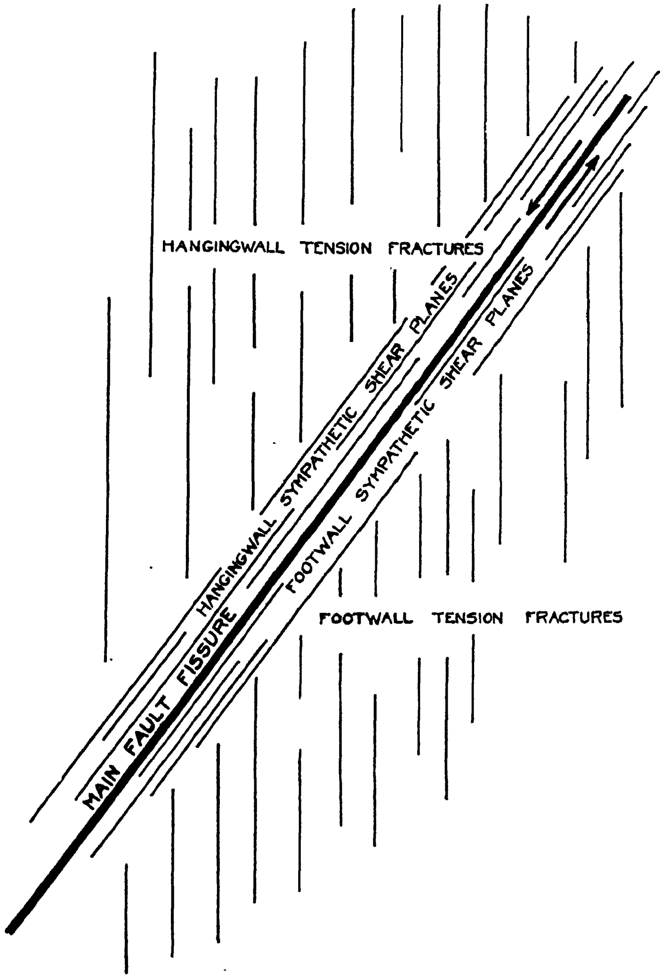

FIG. 4.—IDEALIZED CROSS-SECTION THROUGH VETA COLORADO FAULT-VEIN. NORMAL SHEAR AND TENSION FRACTURES ARE SHOWN.

THE MINERALIZATION

The veins, with one exception, the Palmilla vein, were made by filling of open spaces and replacement of rock along faults of large throw, the intensity at any given place being roughly proportional to the amount of offset. Thus, the most important shoot on the Veta Colorado, El Verde, is in the part of the vein where the fault fissure has the greatest

known offset. Even the minor veins are in faults that have dip slips of at least 50 m. (150 ft.).

The fault fissures appear to have started in premineral time, but the microstructure and texture of the ore indicates that continuous readjust-

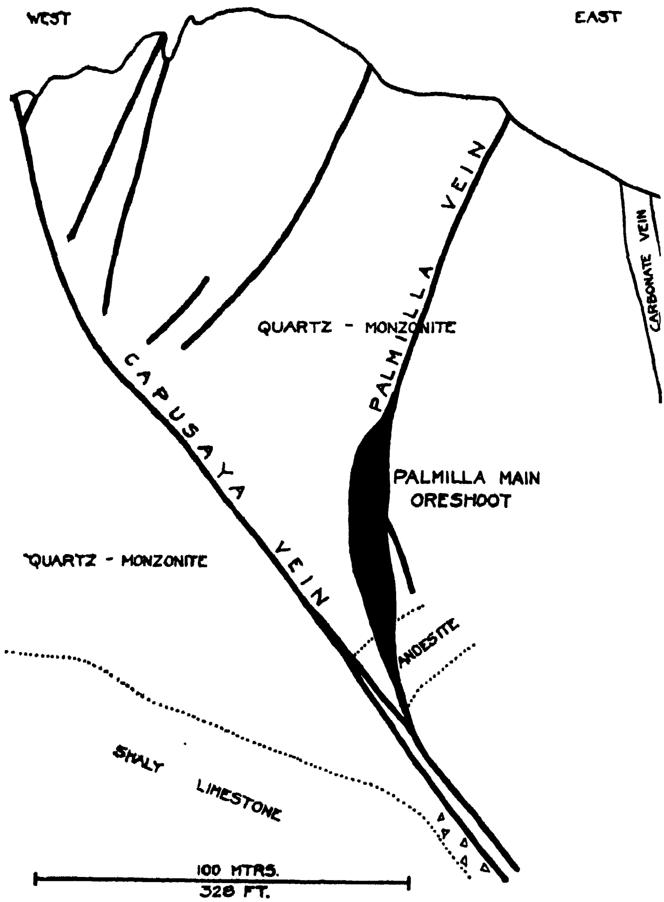

FIG. 5.—EAST-WEST CROSS-SECTION OF PALMILLA AND CAPUSAYA VEINS SHOWING WIDEST PART OF PALMILLA ORESHOOT.

Most of section is from a private report by Rogers, Mayer and Ball, New York City.

ment took place during mineralization. That the movement continued to postmineral time is indicated by gouge-filled seams and water courses included in, and parallel to, the veins.

Location of Veins and Their Relative Importance

The location of the veins is shown by the map of the area (Fig. 1). Of first importance is the Veta Colorado, which is a normal fault fissure

that can be followed approximately 7 km. (4.5 miles) along the strike. The important mines along this vein, listed consecutively from north to south, were the Sierra Plata, Plata Verde, Alfareña, Preseña, and Moreña. None of these are being operated (November, 1929).

The Las Cruces-Cabadeña fault fissure splits off from the Veta Colorado at the Moreña mine, whereas the Veta Colorado fault continues south and contains the footwall vein (Capusaya) of the Palmilla mine. The country between the Capusaya-Palmilla area and the Las Cruces-Cabadeña vein is a horst that is cut by several minor veins, the most

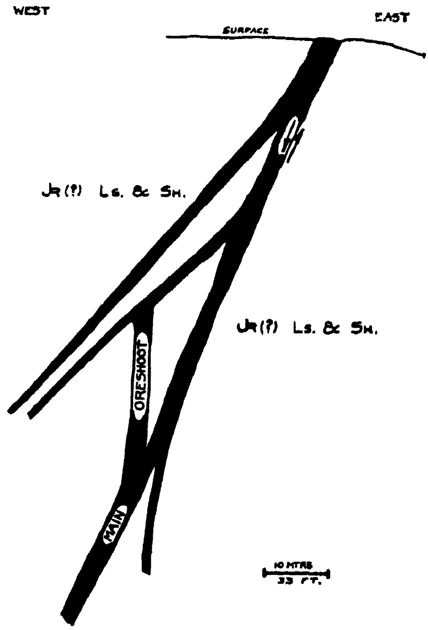

FIG. 6.—EAST-WEST CROSS-SECTION OF SAN JUANICO VEINS AND MAIN ORESHOOT

important of which is the San Juanico. None of these veins are being mined at this time.

The Refugio system of veins is about 160 m. (500 ft.) west of the Veta Colorado and Las Cruces-Cabadeña vein junction. It is small, but important, although at present inactive. The most important shafts are the Refugio, Sierra Madre and Santa Aña.

The Palmilla mine is practically mined out, although it produces occasionally as small leases are granted.

The San Patricio vein, a normal fault, parallels the Veta Colorado on the east but dips west. Inasmuch as the Veta Colorado vein-normal-fault dips east, the intervening block is a graben (Fig. 2). The San Patricio mine has been a small producer but is now idle.

The other important veins occur in the environs of Parral and include the Prieta-Tajo, the most important, the Aguilereña, and the Santa Rosalia-Iguana. All of these are active except the Aguilereña. The Prieta-Tajo mine is one of the richest and most important in Mexico.

Many veins of minor consequence are scattered throughout the area, the majority being north of the monzonite stock. None of these are being worked.

General Features of the Mineralization

Minerals

The known vein minerals of the Parral area are listed in Table 1 in their approximate order of abundance.

TABLE 1.—*Known Vein Minerals of Parral Area*

HYPOGENE

Ore	Gangue
Galena	Quartz
Sphalerite	Fluorite
Pyrite	Chalcedony
Specularite	Barite
Marcasite[a]	Chlorite
Argentite	Adularia
Proustite[a]	
Chalcopyrite	
Tetrahedrite[b]	
Arsenopyrite	

SUPERGENE

Marcasite[a]	Kaolin
Proustite	Chlorite
Hematite	Chalcedony
Chalcopyrite[a]	Carbonates
Manganese oxides	Opal
Argentite	Quartz
Copper carbonates	
Oxidized lead minerals (especially plumbojarosite)	
Oxidized zinc minerals	
Silver haloids	
Native silver	
Native gold	

[a] Classification as hypogene or supergene uncertain.
[b] Identity not proved.

Silica is by far the predominating gangue mineral. It usually averages more than 50 per cent. of the primary vein material of the area. It occurs as well-crystallized quartz, chalcedony, jaspery material, silicified wall rock and a little opal. Study of thin sections indicates that

a little of the quartz and chalcedony and probably all of the opal is supergene.

The fluorite gangue is second only in importance to the silica minerals. It makes up 12 to 15 per cent. of the Veta Colorado vein material. Commonly it is seen with the naked eye, although microscopical examination alone reveals its true importance. Always intimately associated with the sulfides, very likely it has a close genetic relationship to them.

Barite, in places, occurs in relatively large amounts throughout the area. It is not characteristically associated with the sulfides, as the fluorite is, but rather occurs most abundantly in the barren parts of the veins. In the ore itself it appears to have been universally early in the paragenesis because the characteristic bladed forms appear as quartz and specularite pseudomorphs.

Sphalerite is a fairly important sulfide averaging about 3 per cent. in the primary ore of the Veta Colorado. The last few years it has been recovered by flotation at Veta Grande. It is much more important in the Prieta and Tajo mines at Parral, averaging as high as 17 per cent. in large tonnages of ore.

In most of the mines the blende is the amber colored variety, although at Prieta and Tajo mines it is "black jack," the iron-rich variety. Microscopical examination of the tailing from the flotation mill of these mines and of table concentrates made from the tailing showed white sphalerite, which is presumably iron-free. That it did not float more readily than the "black jack" is curious, for the latter variety is supposed to be the least amenable to flotation treatment.

Thin sections show that the "black jack" is not true marmatite, but that the iron is present in iron sulfide (pyrite?), which is arranged crystallographically as dots and bands in the white sphalerite. There seems to be every gradation from white sphalerite with no pyrite (?) inclusions to sphalerite closely packed with them.

One is almost willing to believe that the dark sphalerite was progressively cleansed of its iron by the mineralizing solution.

The sphalerite appears to be one of the silver carriers of this district, but so far no silver mineral has been recognized in it. Some silver is possibly in solid solution in the sphalerite. Assays of rather pure blende always show much silver, sometimes up to 1500 g. (50 oz.) per metric ton. The zinc flotation concentrate of the Prieta-Tajo mill when examined microscopically seems to be almost pure sphalerite, yet on the average it carries about 600 g. (20 oz.) of silver per metric ton.

Galena is found in all the oreshoots of the district but is most important at the Prieta and Tajo mines, where the composite mill head for a month in 1925 showed 6.6 per cent. lead equivalent to 7.6 per cent. galena. The other mines of the district average much less than this, estimated around 2.0 per cent. galena. Locally in some mines, especially

near the bottoms of the shoots in metallogenetic zone C, lead averages 10 per cent. for entire stopes. There is much high-grade lead ore in the Refugio group of mines, which, however, is in zone B.

The galena is usually an important silver carrier. Microscopically, upon etching with HNO_3, it nearly always shows small blebs of a light gray mineral, presumably argentite. Nearly all of it assays high in silver. Some specimens of relatively clean mineral were assayed and gave the results shown in Table 2.

TABLE 2.—*Galena Content of Parral Ores*

	Pb, Per Cent.	Zn, Per Cent.	Ag, Grams per Metric Ton
Coarse galena	44.0	10.4	740
Fine galena	58.4	1.3	230
Coarse galena	39.0	7.3	1830
Galena and sphalerite	17.6	21.3	455

In this case it appears that the fine galena was much lower in silver than the coarse. Coarse galena, however, was usually poor in silver in the Cabadeña and other mines.

Pyrite is relatively unimportant in most of the mines of the district. A few important exceptions include the Prieta, Palmilla and San Juanico mines. In the last two it is a gold carrier, as disclosed by milling tests.

Marcasite (?) is associated with rich proustite ore in the Prieta mine. In the Refugio mine colloform marcasite occurs, which is probably hypogene.

Chalcopyrite appears here and there in the ore as: (1) the well known blebs in sphalerite, (2) late veinlets cutting all primary sulfides. In the first case, most would agree that it is primary. In the second, some of it may be supergene.

Specularite, although small in amount, is found characteristically in the Veta Colorado ore. With quartz it has commonly replaced blades of barite. In such cases the quartz makes up the main mass of the pseudomorph, whereas the specularite in minute plates outlines the border of the original barite crystals. Apparently the red jaspery material, so abundant in Veta Colorado ore, owes its color to specularite.

C. P. Berkey noted adularia in one of the wall rocks (andesite) of Las Quijas mine. Three occurrences in thin section were noted by the writer:

1. In andesite wall rock of the Prieta mine near the new shaft associated with noncommercial quartz-carbonate-sulfide mineralization.

2. In ore from the San Juanico mine associated with late fine-grained quartz.

3. In a fault breccia fragment from ore from the Palmilla mine associated with chalcedony.

Adularia is not commonly associated with ore. Of a large number of thin sections from ore, only the one mentioned above from the San Juanico mine showed adularia associated with commercial mineralization. In the four other known occurrences it was found remote from ore.

C. P. Berkey reported garnet on the sixth level of the Palmilla mine some years ago, occurring in small veinlets as the latest stage of mineralization and associated with fluorite. The presence of it in a mine also containing adularia is interesting, because these two minerals are usually considered indices of the two extremes of temperature of mineralization. Possibly it is a product of a weak, but hot, mineralization accompanying the late intrusions of rhyolite.

Argentite is probably the chief silver carrier of most of the veins of the area, although in only a few instances was it positively identified by the writer. As before noted, much of the galena of the area shows microscopic specks of probable argentite.

Recent work by the Salt Lake City laboratory of the American Smelting & Refining Co. indicated that specks of argentite, ultramicroscopic in size, occurred in the Veta Colorado ore. These tests were made on jaspery material that gives up its silver in cyanide only upon grinding to a fineness far below the commercial limit.

Proustite was identified by the writer in only two mines, the Prieta and Tajo. It is the chief silver carrier in the Prieta oreshoot. Ruby silver has been reported from the Palmilla mine but is unimportant there.

In the Prieta mine much of this mineral may be supergene because peripherally it replaces galena and is commonly associated with some marcasite, chlorite, and kaolin.

Argentite and proustite are the only primary silver minerals so far identified in the district, but probably more of minor importance will be revealed by careful work. Many of the polished sections inspected showed specks of minerals not identified but that possibly are silver compounds.

Calcite and other carbonates are not common hypogene constituents of the commercial mineralization. In places veins or veinlets of calcite cut ore, which are, perhaps, late hypogene.

A large "spar dike" is found in the Palmilla mine hanging wall. This is probably a late hypogene vein.

Along the Veta Colorado concentrations of carbonates occur near, but generally below, the ground-water surface. These are presumably supergene.

Paragenesis of the Minerals

The paragenesis of the minerals of the camp is generalized by the chart, Fig. 7. Of special interest is the fact that most of the quartz

is early and that the sulfides and fluorite came in together. In near-by Santa Barbara these three minerals have the same relations.[3]

An episode in the paragenesis that seems of particular interest is demonstrated by one of the thin sections containing the following:

A. A veinlet of chalcedony with a little fluorite.

B. *A* is cut by a veinlet of fine-grained quartz with proportionately more fluorite than *A* contains.

C. *B* is cut by a veinlet which started with the deposition of pure fluorite on its walls and ended with coarse quartz, sphalerite and galena in the middle. This is presumably the commercial mineralization.

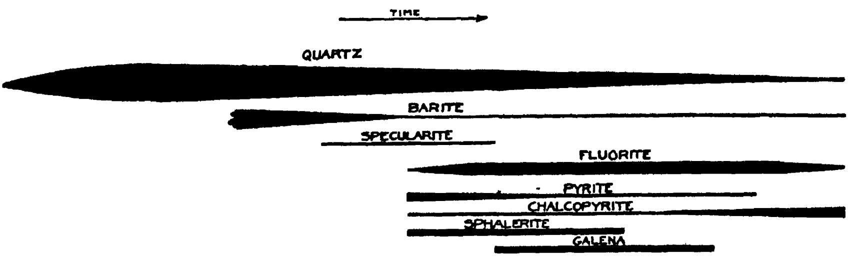

FIG. 7.—DIAGRAM SHOWING PARAGENESIS OF MORE IMPORTANT MINERALS OF PARRAL AREA.

D. *C* is cut by a veinlet carrying chalcedony, proportionately smaller quantities of sulfides and minor amounts of fluorite.

The host rock is andesitic material.

These data check previous observations made regarding the close association of the sulfides and fluorite. Further, they definitely show that some of the chalcedony is primary. The fact that chalcedony was early, followed by a stage during which coarse quartz was deposited, subsequent to which chalcedony was again deposited, suggests that fluorite, because it occurs most abundantly with coarse quartz, caused a silica sol to crystallize as quartz instead of chalcedony.

Depth and Temperature of Origin of the Valuable Mineralization

Nearly all the oreshoots of the Parral area bottom or show a tendency to bottom near the Jurassic (?)-Tertiary unconformity. Because the average original thickness of the Tertiary volcanic rocks is estimated at 3000 ft., and the mineralization occurred after their deposition, 3000 ft. is taken as the probable maximum depth of origin of the oreshoots.

The temperature of origin of the valuable mineralization varied from moderate to low, depending upon the vertical or horizontal position relative to the source, the Parral intrusive being the "epi-centrum," so to speak. The presence of abundant primary chalcedony in some of the veins suggests that their temperatures of origin were not extremely high,

[3] H. Schmitt: *Op. cit.*

whereas the occurrence of massive sulfides in others indicates that temperatures sometimes approached "moderate."

Metallogenetic Zones

The mineralization of the area shows zoning of types around the Parral intrusive (Fig. 1). Four zones, *A*, *B*, *C* and *D*, are recognized, each zone having fairly well-marked characteristics.

Zone *A*, nearest the intrusive, has veins of exceptionally high silica content, abundant pyrite containing gold, and markedly low lead and zinc sulfides. The Palmilla and San Juanico veins are good examples of this mineralization.

Zone *B* is characterized by veins with more lead and zinc sulfides and less pyrite than zone *A*. Gold almost disappears, whereas silver remains about the same. Important veins of this zone are the Refugio group, Las Cruces, Cabadeña, La Prieta and Tajo.

Veins of Zone *C* have less of the base metals. Gold is practically absent and silver is the chief valuable metal. The important veins of this zone are the Veta Colorado, San Patricio and Santa Rosalia.

Zone *D* is distinguished by barren veins with abundant barite and carbonates, but with a very low silica content.

The Veta Colorado, San Patricio and Recompensa veins grade from mineralization of type *C*, on the south, to mineralization of type *D* on the north. Along the Cabadeña vein this same change occurs in a southerly direction.

In all the zones silver is usually the chief valuable metal.

At the outcrop the Veta Colorado is in zone *C*, but in depth it has characteristics of zone *B*, because the proportion of galena and sphalerite increases somewhat. The Prieta oreshoot also seems to grade from zones *C* to *B* in depth; but the Tajo, which is nearer the axis of the intrusive, is entirely in zone *B*.

Small veins, transition types between zones *A* and *B*, occur just north of the Palmilla mine. They are characterized by the abundant sulfides of zone *B* but carry the gold of *A*.

Supergene Alteration

The elevation of the ground-water surface varies from about 1775 m. (5680 ft.) at the Sierra Plata mine to 1700 m. (5450 ft.) at the Prieta mine, with an average for the district of about 1735 m. (5570 ft.). The average depth from the surface is about 100 m. (328 ft.).

The ground water of the district circulates freely, and pumping in any given mine quickly affects the water level in neighboring mines.

In general, thorough oxidation extends a little below the ground-water surface in this area. Locally, along fractures and faults, it penetrates more deeply.

Oxidation resulted mainly in the leaching of sphalerite and chalcopyrite and the loss of most of the zinc and copper. It developed the characteristic oxidized products from pyrite and galena and, on the

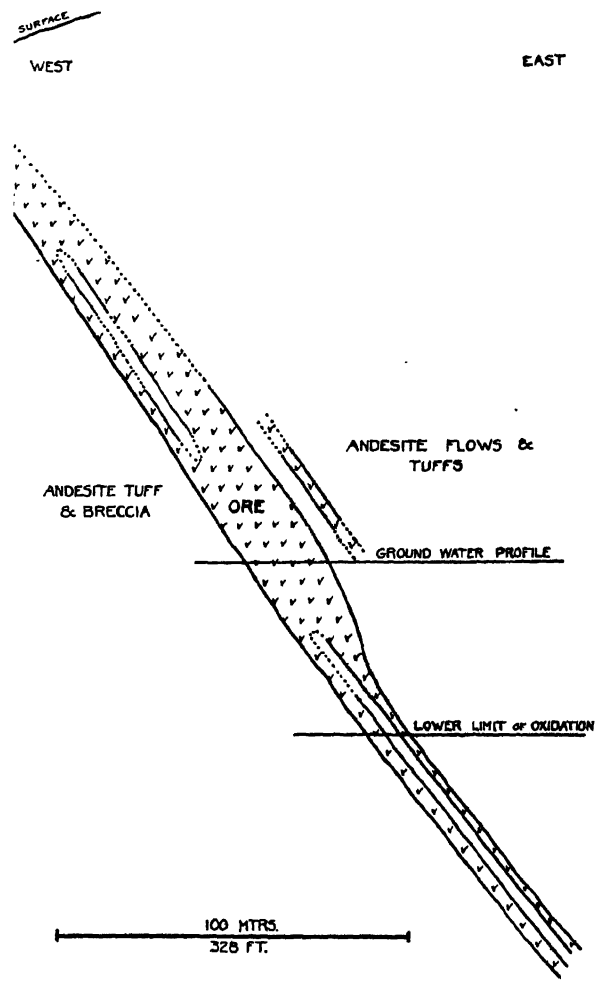

FIG. 8.—EAST-WEST CROSS-SECTION OF SIERRA PLATA ORESHOOT.

Veta Colorado, converted the andesite hanging walls, lying adjacent to ore, to masses of soft claylike ground that is so heavy that square set timbering is necessary to hold it; and even then it caves after a few months.

The footwall on the Veta Colorado, because of its sharp, tight, and steep slope (53° dip), resists oxidation and kaolinization. Most of the permanent haulageways are driven in this wall.

Supergene Enrichment

That supergene enrichment is important in this district is indicated by the following:

1. Most of the very rich ore, where the grade was 10 kg. (330 oz.) of silver or more per ton, was oxidized ore.

2. The rich shoots tend to diminish greatly in size and richness just below the ground-water surface. A good example of this is shown by

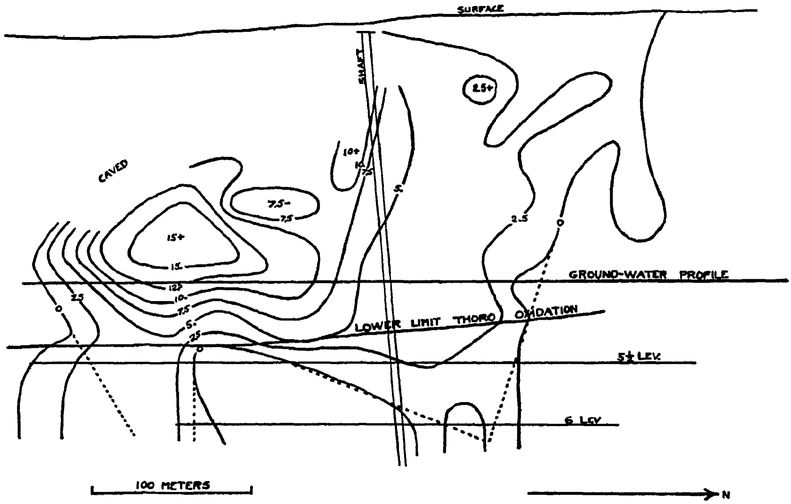

FIG. 9.—EQUAL-ORE-WIDTH LINES ON LONGITUDINAL-VERTICAL PROJECTION OF SIERRA PLATA ORESHOOT ON VETA COLORADO. INTERVAL, 2.5 METERS.

the equal-ore-width chart and cross-section of the Sierra Plata shoot (Figs. 8 and 9).

3. Secondary silver haloids are abundant and make the very rich *brosa* ore.

4. Probable supergene proustite is abundant in the Prieta mine oreshoot.

Hypogene Wall-rock Alteration

The walls adjacent to the oreshoots are always silicified, and probably always propylitized, but neither type of metamorphism is customarily widespread.

Most of the silicification is close to the oreshoots, whereas the less important propylitization is usually farther out. At a surprisingly short distance from the veins, in places only a few feet, in the footwall particularly, is present apparently fresh andesite tuff or other volcanic rocks.

Palmilla hill, however, which contains the Palmilla mine, has been strongly silicified over wide areas, and the country rock around the

Prieta-Tajo vein shows propylitization that extends for more than 100 m. (328 ft.) away, the silicification being relatively near the shoots.

Shape of Oreshoots

The ore of the Parral area is confined to rather definite shoots, generally more sharply outlined than those of the Santa Barbara area. Often they have roughly the shape of a V, showing constant decrease in strike length with depth until they end in one or more roots (Fig. 9.) In the oxidized zone, oreshoot boundaries are in places commercial ones, varying with the cost of extraction and price of metal, whereas in the sulfide zone the ore limits are well defined. The footwall boundary, even in the oxidized zone, is nearly always sharp and coincides with the main fault fissure. Occasionally, however, the footwall is invaded by the rich secondary *brosa* ore, which penetrates below the main fault fissure.

Localization of Oreshoots

Most of the veins are the result of mineralization along normal faults of relatively great offset, but the cause or causes for localization of oreshoots along the strike of the veins is not always clear. Vein junctions seem to have caused ore localizations in the Palmilla mine area, but along the Veta Colorado and other veins there are few vein junctions, and yet the ore was segregated into many definite shoots. Three possible causes for segregation are considered:

1. It is well known that hot springs tend to occur in the valleys rather than in the more elevated places. Mineralizing agents rising along a fault may thus possibly be localized by valleys along the fault outcrop.

2. Cross fractures may have influenced localization, but so far none have been discovered associated with shoots. Such fractures, however, may have been masked by the hypogene and supergene alteration.

3. Along the Veta Colorado the strength of mineralization appears to be more or less proportional to the amount of brecciation occurring along the main fault fissure. All hypogene ore seems to have replaced fault breccia, whereas the barren intershoot areas in the plane of the vein are tight shear zones, not brecciated. There is a suggested relationship between the breccia-ore areas and concavities in the footwall, on the one hand, and tight unbrecciated barren areas and convexities of the same wall, on the other.

It seems reasonable to suppose that the concave areas on the footwall, because of the smaller hanging-wall load caused by a bridging or arch effect, should be areas of greatest brecciation; whereas the convex areas, because of the greater hanging wall load, should be areas of narrow tight shear.

Depth Extension of Oreshoots

The majority of the oreshoots of the Parral area bottom or tend to bottom at about 1700 m. (5500 ft.) above sea level, as John G. Barry of El Paso pointed out several years ago. There are several important exceptions to this, including the Prieta and Tajo mine shoots at Parral, and one of the stronger shoots, the Plata Verde, on the Veta Colorado.

In this district, if the necessary data are available, each shoot ought to be judged on its merits for extension in depth. The strength of the shoot on the lowest level should be given more weight than any regional horizon of apparent bottoming.

Equal-ore-width lines (Fig. 8) and equal-ore-value area charts are particularly valuable for judging possibilities for oreshoot extension.[4]

Surface Criteria for Oreshoots

1. The oreshoots of the Parral area, as before mentioned, are accompanied by strong local silicification. This silicification almost invariably "holds up" the topography, and so most of the mines are at relatively high elevations. The silicification, and thus the elevation, is almost directly proportional to the commercial importance of the shoots. The best mines, those along the Veta Colorado, the Palmilla and the Prieta-Tajo, are on the highest hills. Further, when the oreshoots are examined in detail, it is found that the individual shoots stand out topographically and that the intershoot areas are loci for arroyos.

The Prieta-Tajo mine is on a high hill in the city of Parral. The two shoots, the Prieta and the Tajo, each make a separate knob on the hill (Fig. 10).

2. The close association and paragenetic relation of the fluorite to the valuable mineralization has been discussed. Obviously fluorite is an important criterion for the sulfide mineralization, especially because it is resistant to weathering and remains in the outcrop long after the sulfides have disappeared.

3. Experience has shown that the surface expressions of outcrops of all the known shoots assay some silver, although it may be minute in amount and not make ore. The absence of silver in an outcrop must be considered unfavorable in this area.

4. Some galena or its oxidation products usually occurs in the shoot outcrops. The oxidized lead minerals are usually complex yellow ones including plumbojarosite.

5. Little gossan is found because the pyrite content of the veins is low and the sphalerite is relatively iron free.

[4] H. Schmitt: Extension of Oreshoots with Comments on the Art of Ore Finding. *Trans.* A. I. M. E. (1929) 318.

6. Voids in the quartz-fluorite gangue make significant criteria for sulfides.

7. By studying the metallogenetic zoning around the Parral intrusive it is possible to draw certain outer limits beyond which it is improbable commercial mineralization exists.

Barite, although generally found in the oreshoots, also occurs in the outer barren zone and is an unsafe criterion for the valuable mineralization.

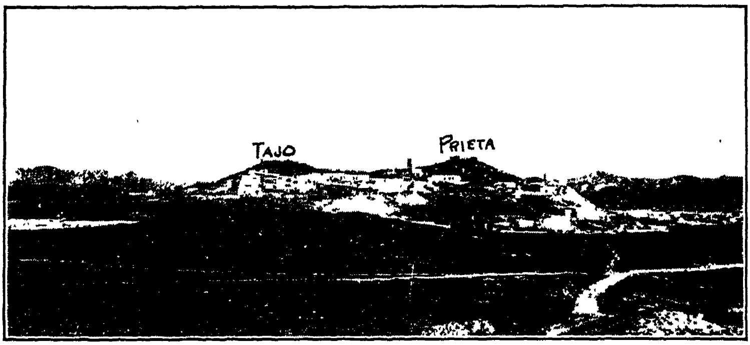

FIG. 10.—SURFACE PLANT, PRIETA-TAJO MINE. PART OF CITY OF PARRAL ON EXTREME RIGHT.

Prieta mine and mill in middle ground. Surface rock in foreground is decomposed intrusive monzonite. In background, volcanic rocks outcrop on higher ground. Two knobs on hill directly back of mill buildings and shaft are outcrops of two main oreshoots; Tajo on the left, Prieta on the right.

Favorable and Unfavorable Outcrop Criteria.—To summarize favorable and unfavorable outcrop criteria for oreshoots in the Parral area:

Essential:
- Silicification.
- Some silver in the outcrop.

Favorable:
- Prominent outcrop, high topographic position.
- Fluorite.
- Yellow, lead compounds.
- Galena.
- Voids.
- Gossan.

Unfavorable:
- Abundant carbonate and barite.
- Location outside of the commercial metallogenetic zones.

SUMMARY

Jurassic (?) limestones and shales were folded, uplifted and then eroded to a mountainous topography upon which a thick series of volcanic

rocks was deposited. This latter event was accompanied by intrusions of magma culminating in the Parral monzonite stock. Block faulting was initiated toward the last of the volcanic epoch because of the continued extravasation of lava.

The Parral stock was probably the center of the volcanism of the area.

Mineralizing solutions followed the larger faults and the vein matter seems to have been deposited before adjustment by faulting was complete.

Distinct metallogenetic zoning can be discerned, the Parral stock being the epicentrum of it.

The mineralization was a very siliceous type carrying quartz, chalcedony, jasper, and fluorite as the most important gangue minerals, and sphalerite, galena, pyrite and argentite as the predominating metallic minerals.

The oreshoots along the Veta Colorado were possibly localized by breccia-filled depressions in the footwall of the Veta Colorado fault.

The oreshoots in the Palmilla mine area seem to be related to vein junctions. The main Palmilla shoot resembles the Comstock lode in structural relations and type of mineralization.

Outcrop criteria for oreshoots in depth in this area are silicification, residual flourite, residual oxidized lead minerals, voids and the presence of some silver.

Acknowledgments

The author is greatly indebted to the American Smelting and Refining Co. for permission to publish the data collected from its properties. While this material was being prepared, valuable aid and criticism were afforded by Dr. W. H. Emmons and Dr. F. F. Grout, of the University of Minnesota. Dr. B. S. Butler, of the University of Arizona, gave pertinent help on the final manuscript.

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DISCUSSION

(R. J. Colony presiding)

H. E. McKinstry, Timmins, Ont. (written discussion).—Mr. Schmitt's description of the Parral district is admirably terse and accurate. As to the causes of locali-

zation of oreshoots, I should be inclined to place even more emphasis than he does upon the influence of vein junctions. Regarding the Veta Colorado, I am not qualified to speak, but many shoots in the district in addition to the conspicuous Palmilla are at vein intersections; for example, in the San Patricio mine, the main oreshoot is at the intersection of a vertical hanging-wall vein with the main west-dipping one, and the Campanas, Plomosas and San Nicolas shoots are at similar junctions. In the Europa mine a number of small shoots are at the junctions or more northerly branches with the main northeast vein zone. The great Tajo orebody is in an angle between two intersecting veins.

A more theoretical but very interesting point is that the whole Tajo-Prieta-Santa Rosalia group of mines is near the point where the northeast-striking vein zone is intersected by the extension of a northwestward-striking system (not shown on Schmitt's map) extending through the Marietta and Carnicerias for some four kilometers to the Providencia and Benito Juarez properties.

The ore deposits of the Parral district, varying as they do from heavy sulfide lead-zinc ores to siliceous silver ores, form a connecting link between the base-metal ores of central Chihuahua and the silver veins of the Sierra Madre. All are characterized by low copper content, low to moderate gold, relatively high silver and sparse arsenic and antimony—chemical characteristics suggesting a consanguinity that is borne out by the regional similarity of igneous rocks accompanying them. The veins of siliceous silver ore are numerous throughout the Sierra of western Chihuahua and Durango but are developed best farther south, in Guanajuato and Pachuca. They are usually in extrusive andesite but when underlying sediments and intrusives are exposed, the veins if not the values are found to continue.

Niggli[5] lays great stress upon the distinction between ores of plutonic and volcanic formations, implying that the solutions giving rise to the latter differentiated from the magma after its extrusion—that is, came directly from the enclosing flows. Certainly the veins in the Tertiary lavas of Mexico and, one might add, the cordillera of North and South America generally, do not support this implication but point to a deeper source. Yet the habitual occurrence of these veins in andesitic flows must be more than a coincidence. Probably it is the result in part of the chemical and physical qualifications of this rock to act as host and partly of the presence of its parent magma lying and cooling at a moderate depth below the surface. In this connection, Schmitt's suggestion that the Parral stock is a volcanic vent is of great interest. My own observation that the Parral monzonite consists of at least three petrographically distinct types points to a reopening of the vent by successive intrusions, one or all of which may have reached the surface.

H. Schmitt (written discussion).—As I understand it, Mr. McKinstry's studies in the district were in the east and southeast; sections in which I have had no opportunity for detailed study. His emphasis on vein junctions as a cause for ore localization is no doubt well taken and recalls the fact that in the near-by Santa Barbara area vein intersections localize richer and wider ore.

Mr. C. L. Baker of Houston, Texas, directs my attention to the fact that Dr. J. Friedlander has found fossils in the close folded Parral sedimentary rocks for which I proposed the name "Santa Barbara series." Friedlander's fossils indicate that part, at least, of the series is Middle Cretaceous; not Jurassic, as I supposed. Friedlander's data will be published in Dr. Carl Burckhardt's book on the Mexican Cretaceous, now in press.

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[5] Niggli: Ore Deposits of Magmatic Origin. Translation by H. C. Boydell, 31 *et seq.* London, 1929. T. Murby & Co.

Ore Deposition in Open Fissures Formed by Solution Pressure

By Alfred Wandke,* Guanajuato, Gto., Mexico

(New York Meeting, February, 1930)

The problem of vein formation has been of particular interest to the writer for years. As his work for a long time was confined largely to districts showing large deposits of copper ore, it was natural that the efficiency of replacement, as one if not the chief control of the emplacement of the ore, seemed a particularly attractive hypothesis. Of late years, in almost daily underground work in the Guanajuato district, he has collected evidence which makes it certain that in a number of cases the ore solutions were under sufficient pressure to force the rock walls apart and to hold them apart, thus offering an open space for the deposition of vein and ore-bearing material.

This paper will concern itself with conditions frequently met with in hypogene ore deposits of the vein type, and will deal in particular with ore occurrence in precious-metal veins of the type classified by Lindgren[1] as epithermal deposits, or metalliferous deposits formed near the surface by ascending thermal waters[2] and in genetic connection with igneous rocks.

To account for the emplacement of this type of ore deposit various writers have suggested the following agencies:

1. Shattering of the wall rocks.
2. Pressure of growing crystals.
3. Magmation, as treated at length by Spurr.[3]
4. Replacement.
5. Openings produced by folding.
6. Shearing due to gravity.
7. Shrinkage produced by the passage of pre-ore solutions.
8. Contraction joints produced by lateral secretion.

To this list there may be added another agency, a discussion of which forms the basis of this paper:

9. Pressure of the ore-bearing solutions.

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[1] W. Lindgren: Mineral Deposits, 3d. Ed., 516. New York, 1928. McGraw-Hill Book Co.

[2] It seems to the writer that it would be well to substitute the word "solutions" for the word "waters," since gaseous solutions would appear to have been necessary to account for some of the indicated mobility and alteration effects of the ore-bearing medium.

[3] J. E. Spurr: The Ore Magmas, 1st Ed. sec. imp. New York, 1923. McGraw-Hill Book Co.

District under Consideration

It is unnecessary to discuss here each of the agencies in detail, as the literature already contains the arguments of the various proponents. Furthermore, to present the subject matter more concretely, attention will be confined almost entirely to the ore occurrence in the Guanajuato mining district. The general geology and ore occurrence of this district[4] have already been presented elsewhere. It may be well, however, to set forth briefly the main facts concerning the geological setting of the veins to be discussed.

The rocks of this district are schists, covered in part by Mesozoic andesites and rhyolites. Following the erosion of these rocks, there was deposited a series of Tertiary red conglomerates, locally underlain by some highly reddened andesites, which in turn were covered by volcanic tuffs, rhyolites, andesites, hypersthene andesites, andesite breccias and quartz rhyolites. These Tertiary rocks have a maximum thickness of 1500 meters. A period of folding next set in, concomitant with the intrusion of granite and monzonite. Shearing along the axes of the major folds and rotational faulting opened the way for the ore-bearing solutions. In some instances, rotational faulting produced a maximum vertical difference of 600 m. between the foot and hanging sides of a given fault. Although many of the orebodies outcrop at the present surface, there is an abundance of evidence showing that these were formed under a cover of several hundred meters of rock. It is thus natural to suppose that the ore solutions must have been under considerable pressure at the time they deposited their loads. Underground study of these deposits over the past five years has resulted in the recognition of the fact that, in some cases, the ore-bearing solutions were under sufficient pressure to force apart the rock walls which now enclose deposits of minable ore.

Views of Other Observers

As far back as 1904, Prof. L. C. Graton, as a result of his studies of the gold veins in the Southern Appalachians, expressed his views quite clearly as regards a special case, but apparently the lead he gave at that time was lost sight of for the more alluring agency of replacement. His statement is as follows:[5]

> It will at once be noticed that there is a striking similarity in structure between these quartz veins and the pegmatite dikes of the tin belts. (Compare Figs. 6, p. 50, and 14, p. 102.) It is believed that this similarity is due to the fact that, although the materials are different, the receptacle is the same. In other words, the form and

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[4] A. Wandke and J. Martinez: The Guanajuato Mining District, Guanajuato, México. *Econ. Geol.* (1928) **23**, 1–44.

[5] L. C. Graton: Gold and Tin Deposits of the Southern Appalachians. U. S. Geol. Survey *Bull.* 293 (1906) 60.

position of both the quartz veins and the pegmatite dikes are almost wholly dependent on the structure of the surrounding rocks. The explanation of the structure of the quartz veins is, therefore, the same as that for the pegmatite, which has already been given at some length (pp. 35–37) and need only be summarized here. It is believed that the veins occur along planes which represented places of weakness in the rocks. The interfoliated veins, or those conformable with the surrounding rocks, are irregular because these places of weakness, the planes of schistosity, were uneven, discontinuous, and irregular. The material from which the veins were deposited, because of its probable greater fluidity than that from which the dikes solidified, sought out even more irregularities, and consequently the veins are less regular than the dikes. In short, it is believed that the bodies of quartz which now exist, did not solidify in open spaces of corresponding dimensions which were ready to receive the solutions, pushing their way along what may in many cases have been the merest fractures, but actually forced the walls apart and made the receptacles in which their load was deposited. The force of crystallization [Dunn, J. E., Reports on the Bendigo Gold Fields, Victoria Department of Mines. 1896, p. 25] may have aided somewhat in this expanding of the openings, but it is believed that the principal factor was the pressure under which the solutions reached this zone of deposition. This pressure, which must have exceeded that resulting from the weight of the overlying rocks, was transmitted from a greater depth, where the weight of the overlying rocks was greater than above. An idea of the structure of these infoliated veins is given by Figs. 12, p. 100, and 14, p. 102. The crossveins, though deposited at the same time as the others, are more irregular than the conformable veins, since they occupy later fissures which are definite cracks breaking across the schistosity.

Somewhat later J. E. Spurr, in speaking of the ores of Silver Peak, Nevada, which he studied in 1905, says,[6] "That the lenses are the fillings of cavities, which were present in the schist, is out of the question. The parallelism of the schistosity with the curving walls of the lenses shows that the intrusion filled spaces which it, itself, created." Later Spurr[7] dealt with this subject in more detail and said: "The quartz magma collected in larger masses and by itself made on a small scale an independent intrusion in nearly the same sense as the Alaskite magma had done." That this mechanism for the emplacement of veins has continued to remain open in the mind of Spurr is amply attested to by his later work.[8] While one may differ with Spurr as regards his theory of an ore magma, the clear way in which he has proceeded to demonstrate the necessity for the ore-bearing solution, under certain conditions to open its own way, is worthy of serious consideration by all who are interested in the formation of ore deposits. Thus on pages 153 and 155 Spurr says:

The chief ore-bearing veins have formed mainly by replacements of crushed and sheeted fissure zones; but many of the veins of barren quartz, have evidently not so formed, but are clean types of fissure veins, with no evidence of replacement, and with a sharp and clean line between vein and wall. Of such veins we are accustomed

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[6] J. E. Spurr: Genetic Relations of the Western Nevada Ores. *Trans.* A. I. M. E. (1906) **36,** 394.

[7] J. E. Spurr: Ore Deposits of Silver Peak Quadrangle, Nev. U. S. Geol. Survey *Prof. Paper* 55 (1906).

[8] J. E. Spurr: *Op. cit.* See particularly chapter on The Injection of Mineral Veins.

to think and say that they were deposited in open fissures, although I have already shown that the pre-existence of an open fissure before vein deposition is never probable, and in many cases plainly impossible, for in some cases the vein magma has evidently forced open the rocks dike-wise and made room for itself.

. . . Therefore, as to *the mode of formation of vein*: Some vein-forming solutions, from the pegmatite solutions up through the range to those which have formed relatively near the surface, appear on occasions to be intrusive and form intrusive veins, or vein-dikes.

That there is an intimate relationship between the processes of magma intrusion and the emplacement of veins is well brought out in an interesting summary paper by Dr. C. S. Ross.[9] Dr. Ross also calls attention to the fact that when a solution-rich magma solidifies, the liberated solutions, which later on are to form ore-bearing veins at some distance from the intrusive, may begin their migrations, "with a new and more effective force—a force that the magma may not have possessed even in the early stages."

Even the layman today accepts the theory of magma intrusion. It would seem, however, that geologists as a group have failed to grasp the fundamental fact that ore solutions may exert enormous pressures, and as easily force the rock walls apart as does an ordinary trap dike. It is but natural to question the ability of ore-bearing solutions to lift a load represented by the weight of several thousand feet of overlying rocks. That this problem is being considered may be illustrated by the following quotation:

A vast fund of information on the structural features of veins could be obtained by an intensive study of the pre-mineral dikes in any mineralized district, and a study of the dikes themselves, if exposed, as in mining a vein, would be illuminating. Such study would tend to eliminate some of the strained theories on the formation of veins and ore-deposits. Passing from the dikes to the veins, which are consequent upon the dikes, what of the forces, which are back of mineralized (and metallized) gases, vapors, and tenuous or viscuous fluids, sufficient to inject them well toward the surface zone against the resistance of the superimposed strata?[10]

After classifying the mechanics of vein formation Leverett adds:

The true mode of origin of many veins has apparently been overlooked—namely, the repeated reopening of the vein fissure by fault movements acting contemporaneously with mineralization, and the repeated filling of the small openings as produced. To designate this mode of formation the term vein formation by secretion has been proposed.

Another recent view regarding ore genesis is contained in a paper by Dr. Hulin,[11] who gives the following list under Mechanics of Vein Formation:

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[9] C. S. Ross: Physicochemical Factors Controlling Magmatic Differentiation and Vein Formation. *Econ. Geol.* (1928) **23**, 864–886.

[10] S. R. Leverett: The Man at the Face and Ore Genesis. *Eng. & Min. Jnl.* (1929) **127**, 645.

[11] C. D. Hulin: Ore Genesis and Ore Shoots. *Eng. & Min. Jnl.* (1929) **127**, 317–320.

1. By filling of an open fissure.
2. By replacement.
3. By force of crystallization of vein minerals.
4. By hydrostatic pressure of vein-depositing solutions.
5. By injection of a viscous ore magma.

While Dr. Hulin evidently considered the hydrostatic pressure of vein depositing solutions sufficiently important to be listed as one of the agencies of vein formation, it is unfortunate that he did not discuss this subject at greater length.

That not all recent thinking has shown a trend toward accepting the theory that ore solutions can force apart the rock walls is rather conclusively shown in the following quotation from a paper by Dr. Whitman.[12] On pages 331 and 332, he says:

> That metasomatism is the general rule in the formation of ore deposits is seen in the facts (1):—That cases where the walls of orebodies have been thrust aside by the injection of the orebody are rarely if ever demonstrable. (2):—That the cases of consistent crustification are also rare. (3):—That if ore and gangue or vein matter are removed from their matrix by mining or imagination, the remaining open space by no stretch of imagination could be conceived to have existed prior to the ore nor to have been created by displacement and (4):—That in the majority of ore deposits all stages of transition from country rock to ore can be found, either in the main orebodies or in the weak associated mineralization.
>
> In most cases the orebodies contain sheets, or sheet remnants, of wall rock, or else other fragmental inclusions of the wall rocks oriented parallel to the walls or to the wall structures or clearly transitional into the wall rock by the gradual dwindling of mineral or gangue outward from the central region of the orebody.
>
> In the associated non-commercial mineralization, therefore, one is to look for the key to the formation of the neighboring bonanzas. In such places ideas of injection and crustification fade into mere fantasies.

In the light of the observations of the men who have expressed a belief in the efficacy of solution pressure as an agency for the emplacement of orebodies, the statement of Dr. Whitman presents a rather discordant note. In the author's own observations (pages 10 to 13), it will appear that the things Dr. Whitman believes cannot happen have apparently taken place.

Power of Solutions

Experimental proof is not lacking that a solution may have the pressure required to force apart rock walls, and thus form an opening where orebodies may develop. In 1922, Dr. G. W. Morey,[13] of the Geophysical Laboratory, wrote a paper dealing with the development of pressure in magmas. He says:

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[12] A. R. Whitman: Diffusion in Ore Genesis. *Econ. Geol.* (1929) **24**, 330–335.

[13] G. W. Morey: *Jnl.* Wash. Acad. Sci. (1922) **12**, 219–230.

The eutectic between $K_2Si_2O_5$ and SiO_2 lies at the remarkably low temperature of 520°. If a mixture of K_2OSiO_2, and H_2O, containing 9.1 per cent. of H_2O with the other ingredients in the molecular ratio $SiO_2/K_2O = 4.26$, be cooled from a high temperature, the vapor pressure of the mixture will fall as the temperature falls. The mixture will not begin to freeze until it has cooled to 500°, when crystals of quartz and the ternary compound $KHSi_2O_5$ will separate. The vapor pressure of the solution at this temperature is 160 atm. On further cooling, the substances continue to crystallize and the pressure increases rapidly. When the temperature has fallen 20°, to 480°, the water content has increased to 10.2 per cent. and the pressure to 180 atm. When the temperature has fallen to 420°, the water content has increased to 12.5 per cent., and the pressure to 340 atm., more than double the pressure at 500°.

It is evident, then, as a magma containing water and other volatile components cools with consequent crystallization, the pressure will rapidly rise from its initial value, and as the cooling continues the pressure will increase until the temperature of maximum pressure has been reached, or until the pressure is relieved by escape of the volatile material. In the first case, which is that in which the liquid cools under a crust of sufficient weight and strength to withstand the internal pressure, the liquid will solidify as an intrusive mass. In the case of an actual magma the fact that water has a critical temperature of 374° C. has no significance, because of the probability that enough material will remain in the solution to raise the critical temperature of the mixture the requisite amount. The water, containing in solution residual material such as dissolved gases, boric acid, sulphur, and probably some alkalies, will be available for metamorphic processes.

This increase (in pressure) is rapid whether measured in terms of decrease in temperature of the three-phase equilibrium or in terms of the content of volatile material in solution. From the latter fact it follows that in systems of the type of magmas, in which the nonvolatile material is composed of such substances as the silicates and in which the pressure required to retain any considerable portion of water in solution must be large, a comparatively small amount of crystallization will result in a large increase in pressure.

The foregoing statements furnish a proof that there undoubtedly exists in every intrusive body a mechanism by which the solutions given off as a result of crystallization may readily attain a pressure that will not only permit them to force apart the walls of the fissures through which the solutions escape, but also, under certain conditions of depth and temperature, actually to force[14] their walls apart in a manner commonly illustrated in dike intrusion. This pressure developed during crystallization may have no connection whatsoever with the intrusive force of the parent magma, but may become effective only after the intrusion of the magma has been halted. The solutions released by crystallization are then free to continue the act of intrusion already begun by the magma, and to carry this forward with a renewed and much more effective force. These facts indicate that the end products of crystallization may play the role of marking the most active stage, as regards vein formation, in the entire intrusive history of the magma.

[14] See also W. Lindgren: *Op. cit.*, 173.

Observations on Flat-lying Veins

In the Sirena mine of the Guanajuato Cons. Mining & Milling Co., the ore occurs in a well-defined vein and also as a stockwork to the hanging-wall side of the main vein. Mining operations in the stockwork are of the cut-and-fill type. Since these cut-and-fill stopes are sometimes open along the ore for 15 to 100 meters, an excellent opportunity is afforded for studying the occurrence of the ore. In one small stope, mining had

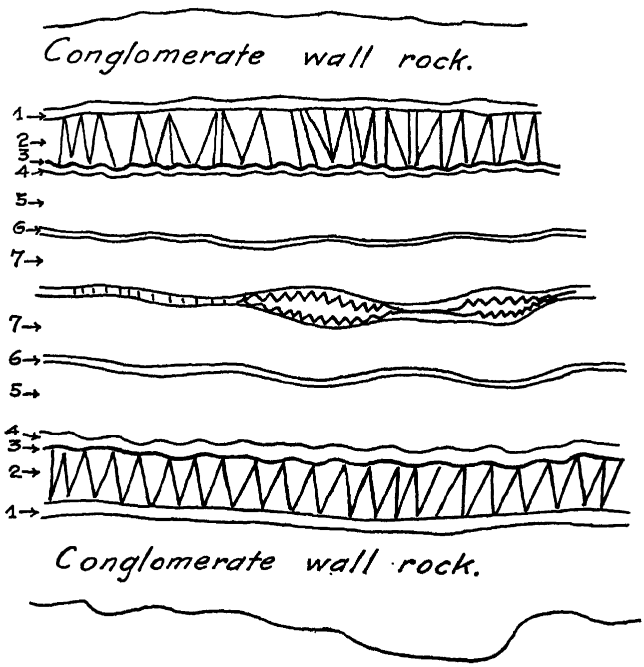

Fig. 1.—Section across amethyst vein.

1. White chalcedonic quartz, 1 cm. wide.
2. Well-crystallized amethyst, 3 cm. wide.
3. Carbonate, about 1 mm. wide.
4. Banded chalcedonic quartz with bluish tinge, 1 cm. wide.
5. White chalcedonic quartz, 2.5 cm. wide.
6. Amethyst, very pale in color, 2 mm. wide.
7. White crystallized quartz with vugs lined with pale amethyst, 1 to 3 cm. wide.

been carried on until the vein had been exposed for a length of 20 m. In the ore being mined, the author's attention was attracted to a well-defined vein showing a particularly noticeable development of amethyst. As the vein was followed along its exposed length, the amethyst bands, forming along both walls, retained their relative distance apart throughout the entire exposed section. Thus, in places the amethystine walls would be only 9 cm. apart, and at others would be separated by from 12 to 15 cm. At no place did the amethystine walls close down to less than 9 cm. So striking was the occurrence that the writer had a large block of the vein

broken out and sent to his office for careful study. A section across the vein shows the mineral developments outlined in Fig. 1.

For weeks it was possible to study this vein underground, until a stope had been carried along the ore, which this vein cuts, making an opening 20 m. along the strike and 15 m. along the dip. This vein dips at an angle of 40° to 45°, and not once were the two amethyst bands nearer than 9 cm. to one another. One of the writer's earliest conclusions had been that the bands of amethyst represented deposition in an open space. It is obvious that such a conclusion demands either that a rather large open space had been produced, which remained entirely open for a considerable length of time to permit the formation of the vein in question, or that the walls were forced apart by the solutions that carried the vein-forming minerals.

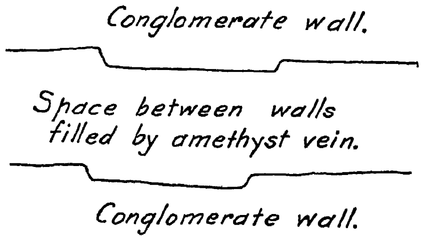

FIG. 2.—ANGULARITIES IN WALLS USED AS HORIZON MARKERS.

In a region where shearing produced pathways which permitted the passage of ore-bearing solutions, the natural conclusion is that the open space in question was produced by one wall of the vein sliding over the other for a considerable distance, leaving an opening because of the failure of the walls to match. Even though the size of the opening produced might militate against such a happening, this is by far the simplest

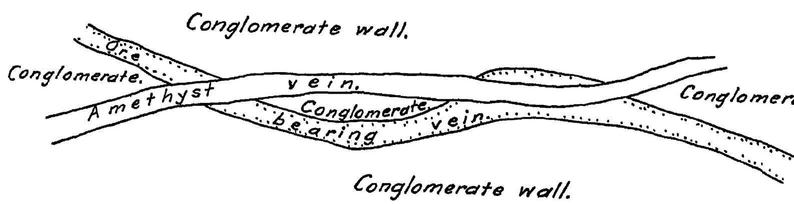

FIG. 3.—ORE-BEARING VEIN CUT BY AMETHYST VEIN.

explanation and must be disposed of before other views can be considered. The first difficulty with this simple explanation is the fact that in a number of instances sharp angularities were observed in one wall which corresponded exactly with those in the other, thus indicating that the amount of lateral movement must have been not more than a few centimeters at the most. This relation is shown in Fig. 2.

The amethyst vein also cuts through an ore-bearing vein that came in usefully as a horizon marker. At one particular place conditions were observed as shown in Fig. 3. Since the displaced parts of the ore-bearing vein showed almost no lateral movement, this vein at once became an

important marker, showing that the only movement of the walls of the amethyst vein was a vertical movement, corresponding to the width of the invading amethyst vein. To produce the vertical movement, it seemed plausible to consider whether or not the force of crystallization might have been sufficient to lift the rock walls. The very fact that the amethyst vein is strikingly banded, the various bands showing no interlocking effect, would seem to indicate that this process could not have produced the results now seen.

A stronger proof against this process is shown by Figs. 4 and 5. From this it is clearly seen that, at the time the amethyst began to form, the vein walls were far enough apart to accommodate rock fragments. Furthermore, after the amethyst had been deposited the walls were far enough apart to permit a fragment of amethyst-coated hanging wall to drop into the open space between the amethyst-coated walls. There are, however, cases where the rock walls seem to have been forced apart by the pressure of the growing crystals. In such examples, almost always narrow veinlets about 2 or 3 cm. wide, the quartz crystals growing from either wall are completely intergrown, a condition not seen in Figs. 4 and 5.

It may also be suggested that the walls were forced apart by a thick jelly-like colloidal solution which acted as a dike. It would seem that the banded structure of the vein, as well as the fact that fragments of rock fell from one of the walls into the open space, not only before vein matter was deposited but also after the amethyst had begun to form, would indicate that such a process had not been operative. One might also argue that the Liesegange effect in a colloidal solution would easily account for the banding, and thus do away with the necessity of having a varying solution passing through the rocks. The near-by walls show conclusively, however, that a varying solution was passing through the rock walls; thus there can be found narrow veins of amethystine quartz, which pass for considerable distances into the walls away from the main vein. These narrow solid amethyst veins are, in turn, cut by veins of white crystallized quartz, entirely similar to material found in the center of the main banded vein. In a near-by stope, where it is evident that considerable movement and adjustment of the rocks took place during the mineralizing process, it is, because of crosscutting relations, possible to establish the following order of mineralization: ore, vein carbonate, chalcedonic quartz having a blue tone, amethystine quartz, white quartz, clear crystalline quartz lining vugs. This order, when compared with that of the main vein, shows conclusively that we are dealing with a mineralizing solution which changed gradually in composition.

It would seem, therefore, that only one conclusion can be drawn; that is, that under certain conditions, the mineralizing solutions were introduced into the rocks under a sufficient pressure to force apart the

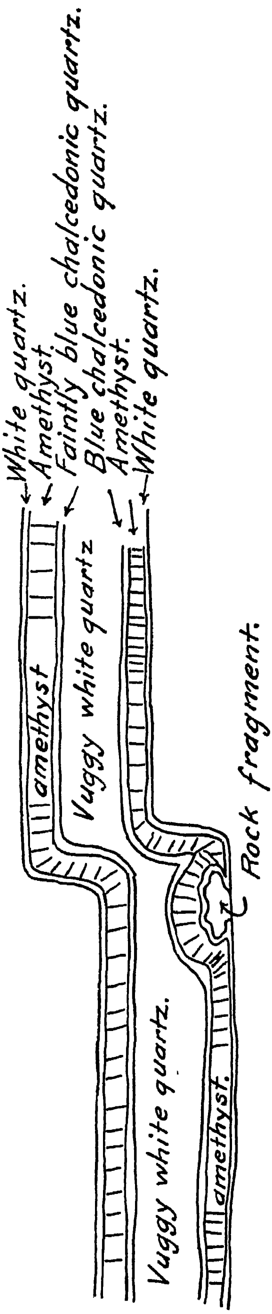

FIG. 4.—ROCK FRAGMENT IN SPACE BETWEEN VEIN WALLS.

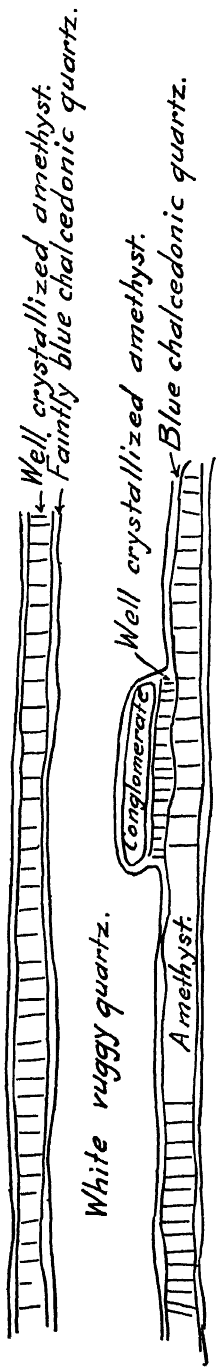

FIG. 5.—INDICATION THAT SPACE BETWEEN AMETHYST BANDS WAS WIDE ENOUGH TO ACCOMMODATE A PIECE OF WALL ROCK COATED ON ONE SIDE WITH AMETHYST.

rock walls and to hold them apart during the entire period in which the vein was being formed.

Another example was available for study for a considerable length of time, since the rock in which the vein occurred was of sufficient grade to permit mining to be carried on. At one end of the stope the conditions depicted at either end of Fig. 6 were observed. A little to the north, the ore-bearing vein, originally about 20 cm. wide, increased in width, by the greater development of barren white quartz, to 1 m. 10 cm. It appears probable that here not only was the hanging wall lifted bodily, but, by an adjustment of stresses, the upper or hanging-wall side of the vein was badly shattered and broken, bringing about the conditions depicted in Fig. 6.

Observations on Vertical Veins

The preceding examples, taken from rather flat-lying veins, would, from the ease with which the observations were made, seem to establish rather conclusively the thesis of this paper—the ability of ore solutions actually to force apart the rock walls.

In vertical veins the proof is more elusive, and it is only where a particularly clear example can be found that observations can begin with any degree of certainty. It has been generally accepted that lenticular openings for orebodies are usually produced by the failure of the vein walls to match, after the rock has been subjected to a

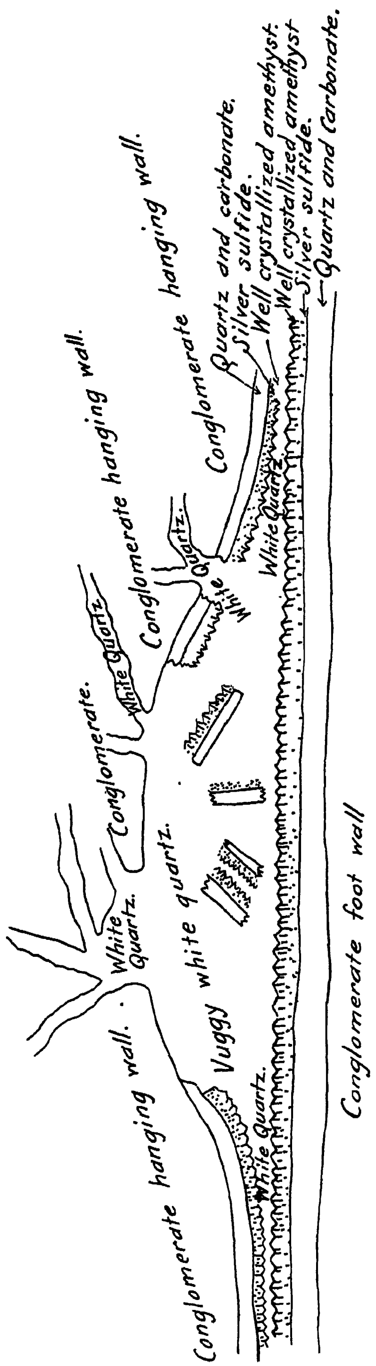

Fig. 6.—Fragments of banded silver sulfide and amethyst with white quartz within.

shearing stress under the influence of gravity. In many cases, it is probable that the failure to match produced the open space that became filled with ore. On the other hand, there are certain practical considerations which seem to indicate that it would be impossible to maintain an opening of the size required to contain any one of some of the smallest minable orebodies of this district. This is particularly true of orebodies that, having well-defined walls, show considerable gouge along one of the walls, indicative of the sliding movement so essential in bringing irregularities into juxta-position and thus producing an open space of the required sort. Mining operations, carried on well below the zone of ground-water circulation, however, have clearly shown how impossible it would have been for an open space to be maintained under the conditions of gravitative settling. It was while trying to work out an easy mining scheme for some of these gouge-walled orebodies that the author began to question the ability of the accepted theory of "failure to match" to explain some of these vertical, or almost vertical, lenslike orebodies.

The facts needing explanation are briefly as follows: The ore of the vertical veins occurs in lenslike bodies which are from 20 to 100 m. long by ½ to 5 m. wide. As a rule the ore is well banded, the same mineral succession being found along both walls. The bands, the loci of the valuable minerals, can be traced throughout the entire length of a given oreshoot and remain persistent for its height. One wall or the other of the oreshoots is almost always badly fractured, so that a cut-and-fill system of mining must be carried on rapidly if caving is to be prevented. Many parts of the veins are frequently crowded with angular, instead of rounded, wall-rock fragments.

The banded, or crustified, structure of the veins, as well as the clearly defined walls, at once suggest that the orebodies were formed in an open space. The heavy gouge, usually found along one of the walls, indicates that considerable movement had taken place, and so the theory of "failure to match" seems a logical explanation of the ore lenses. As mining operations were carried on, it became evident that an opening large enough to contain even one of the smallest ore lenses could never have been formed, as the shearing movement necessary would have maintained constantly a crushed mass of rock between the fissure walls. It has seemed necessary to conclude, therefore, that the opening, now filled with banded, or crustified, vein matter, was produced by the pressure of ore-bearing solutions which, by causing a minimum of movement, nevertheless forced the walls of a preexisting fissure sufficiently apart to permit the formation of the banded orebodies. Such a mechanism would not only account for the emplacement of the stretches of vein that are beautifully banded and free of all rock fragments, but would also account for the presence, in the same portions of the vein, of a jumble of rock fragments, which show a crustification of the ore minerals similar to the

banded vein matter along the walls. The rock fragments, which range in size from minute particles to blocks weighing several tons, are invariably sharply angular—not well rounded, as they should be had they been formed along a shear zone. It appears probable that as the fissure walls were forced apart by the pressure of the ore-bearing solutions, rock material, from the more badly fractured walls, caved into the space between the slowly spreading walls. One is forced to conclude, therefore, that a vein which at first glance would appear to have formed by replacing broken rock material in a fracture zone has in reality been formed by filling an open space produced by the pressure of the ore-bearing solutions.

Summary

This paper directs attention to a process of ore emplacement which seems not to have received due consideration. Experimental work has indicated that ore solutions frequently depart from the parent magma under a pressure that would compel them to force apart the rock walls. Certain examples of flat-lying veins are cited which apparently show conclusively that this process has been operative. It is also concluded that certain vertical lenticular orebodies, where the proof is less conclusive, have nevertheless also been emplaced in openings formed by the pressure of the ore-bearing solutions.

DISCUSSION

(R. J. Colony presiding)

C. B. E. Douglas, Pachuca, Hidalgo, Mexico (written discussion).—It occurs to me that the subject of Mr. Wandke's interesting article might be carried a little further. Rock is slightly compressible. Over long periods the pressure necessary to hold the walls of a fissure apart horizontally at any point may be assumed to be approximately equal to the vertical pressure at that point in the rock due to the height of the rock column to the surface. At a depth of 5000 ft. this would be about 400 atm. if the rock had a specific gravity of 2.7. (Its specific gravity would be slightly greater at a depth of 5000 ft. than at the surface, but the difference would be so slight that it can be neglected.) For the sake of illustration, let us assume that the specific gravity of the solution given off by the cooling magma would be 1.35, or just one-half that of the rock.

If there were solution in a fissure under sufficient pressure to hold the walls apart at a depth of 5000 ft. (400 atm.), at a point 2000 ft. higher in the fissure the solution would have an excess of pressure of 80 atm. above that necessary to hold the walls apart, which would be available for rock compression, extending the fissure, etc., for the rock pressure there would be

$$400 \text{ atm.} - \frac{2000 \times 2.7 \times 0.434}{14.7} \text{ or } 240 \text{ atm. (approx)}$$

and the solution pressure would be

$$400 \text{ atm.} - \frac{2000 \times 1.35 \times 0.434}{14.7} \text{ or } 320 \text{ atm. (approx).}$$

The difference would be lessened by friction if the solution were moving, but the semigaseous nature of the solution would accentuate the difference greatly.

Were the rock over the magma homogeneous, the tendency would be for the solution expelled during the crystallization of the magma to extend any incipient fracture to the surface at the expense of supporting the walls in depth, and the rate at which it would do so would depend upon the rate at which solution were made available as a result of the progress of crystallization of the magma.

It is assumed that the solution would collect in a reservoir over the top of the magma. As the solution pressure in the reservoir at the top of the magma would be relieved, perhaps rapidly, on the opening of an outlet, as up a fissure, it would not be surprising if the amount of solution that escaped might not be sufficient to reach the surface, and the fissure be not extended to the surface either.

If the height of the fissure filled were great the relative increase in solution pressure over rock pressure at the higher elevations might result in a state of equilibrium being reached near the top of the fissure while at some point below, near the magma, the rock pressure might preponderate to such an extent as to close the fissure, thus shutting off the solution above from its source until further crystallization of the magma should have expelled enough more solution to raise the pressure there sufficiently to reopen it again. Then the elasticity of the solution (semigaseous) would cause it to rise in the fissure again with something like explosive violence.

Presumably some crystallization would have taken place on the walls of the upper part of the fissure where it has been filled with the temporarily stagnant solution. Successive upward bursts of solution, if of the same magnitude as the first, would reopen the part where some crystallization had taken place by a similar amount, as before. (Within limits there would probably be a tendency for the compression stresses produced by previous openings back in the walls to become dissipated if the time between solution surges were long enough.) Had crystallization from the solution of the previous surge sealed the fissure in places, it would seem possible that instead of following the center of the previous opening, a succeeding surge might tend to break round the sealed part in places, perhaps tearing off a fragment of rock already frozen to the vein, especially if the original opening of the fissure had caused subsidiary cracks to form.

It is possible that the surge might be so explosive as to shatter a vein that had already practically filled the fissure and so produce that effect of later quartz banding around fragments of earlier quartz that has led to so much discussion.

Also, the explosive effect might upset the normal sequence of crystallization. Various substances might all be thrown down at the same time in a semicolloidal state, later to be incorporated in and become part of crystal aggregates. Or the semicolloidal material might remain in that state through successive explosive surges and only assume crystalline form at some later period, perhaps long after the intrusive action ceased.

When the great differences in physical properties between aqueous solutions, containing perhaps large quantities of gases, or well above the boiling point of water, and ordinary dike magmas are considered, it does not seem sound to treat them as analogous in their effects. Dike magmas would be relatively inelastic as compared with magmatic solutions and of so nearly the same specific gravity as the wall rock that the relation of dike pressure to wall-rock pressure would be much the same at any elevation. Consequently there would be the difference that would result in equilibrium being reached at some point above while the wall-rock pressure below became sufficiently preponderant to close the fissure. As a result, the rise of the dike would be continuous instead of intermittent, and perhaps explosive in part of its course.

Zonal Relations of the Lodes of the Sumpter Quadrangle*

By D. F. Hewett,† Washington, D. C.

(New York Meeting, February, 1931)

Contents

Introduction

In an earlier report on the Sumpter quadrangle[1] it was briefly stated that the quartz lodes of that region might be separated into several groups or belts on the basis of the kind and abundance of the minerals of which they are composed, and that these belts were disposed as areal zones around the principal intrusive mass of the region. When the opportunity to revisit the region was offered in 1929, as a result of a cooperative agreement between the State Mining Board of Oregon and the United States Geological Survey, it seemed advisable to pay particular attention to this feature of the lodes, in the hope that if the idea of zonal relations was confirmed it might serve as a guide in exploitation.

* Published by permission of the Director, U. S. Geological Survey.

† Geologist, U. S. Geological Survey.

[1] J. T. Pardee and D. F. Hewett: Geology and Mineral Resources of the Sumpter Quadrangle. Oregon Bur. Mines and Geol. *Min. Resources of Oregon* (1914) **1**, 66–67.

Since 1914, when the earlier report on the district appeared, much has been recorded concerning the zonal distribution of ore deposits around centers of intrusion in numerous regions, and there has been much speculation concerning the significance and cause of these relations, therefore much more is known now about the hypothesis of zonal relations than in 1907, when it was first proposed by Spurr.[2] Unfortunately, in reviewing the relations of the lodes of the Sumpter district, much smaller parts of the workings of the mines were accessible in 1929 than in 1914, and the necessary records of production are dispersed and obtainable with difficulty, if at all.

Geologic Features of Eastern Oregon

The first comprehensive contribution to the geology of eastern Oregon is that made in 1900 by Lindgren,[3] who, in addition to examining the mines, made a geologic map of an area in eastern Oregon and western Idaho about 50 by 100 miles. The Sumpter quadrangle was studied by Pardee[4] in 1908 and 1909 and by Pardee and Hewett in 1914, and the Wallowa Mountains by Swartley[5] in 1914 and by Ross[6] in 1924. In 1913 Grant and Cady[7] studied the geology and mines near Baker. In 1929, under a cooperative agreement between the State Mining Board and the U. S. Geological Survey, James Gilluly began an areal and economic survey of the Baker quadrangle and D. F. Hewett reexamined the accessible mines of the Sumpter quadrangle.

Geology of the Sumpter Quadrangle

The geologic record of the Sumpter quadrangle reveals most of the recorded outstanding events in the geologic history of eastern Oregon,

[2] J. E. Spurr: A Theory of Ore Deposition. *Econ. Geol.* (1907) **2**, 781–795; also (1912) **7**, 480–493.

W. H. Emmons: Primary Downward Changes in Ore Deposits. *Trans.* A. I. M. E. (1924) **70**, 964–992.

Relation of Metalliferous Lode Systems to Igneous Intrusives. *Ibid.* (1926) **74**, 24–70.

Relations of the Disseminated Copper Ores in Porphyry to Igneous Intrusives. *Ibid.* (1927) **75**, 797–809.

H. Dewey: The Mineral Zones of Cornwall. Geol. Assoc. *Proc.*, (1925) **36**, 107–135.

R. H. Rastall: Metallogenetic Zones. *Econ. Geol.* (1923) **18**, 105–121.

[3] W. Lindgren: The Gold Belt of the Blue Mountains of Oregon. U. S. Geol. Survey, *Ann. Rept.* (1901) Pt. 2, 561–776.

[4] J. T. Pardee: Faulting and Vein Structure in the Cracker Creek Gold District, Baker County, Oregon: U. S. Geol. Survey *Bull.* 380 (1909) 85–93.

[5] A. M. Swartley: Ore Deposits of Northeastern Oregon. Oregon Bur. Mines and Geol. *Mineral Resources of Oregon* (1914) **1**.

[6] C. P. Ross: Unpublished manuscript.

[7] U. S. Grant and G. H. Cady: Preliminary Report on the General and Economic Geology of the Baker District of Eastern Oregon. Oregon Bur. Mines and Geol. *Mineral Resources of Oregon* (1914) **1**, No. 6, 131–161.

and the ore deposits explored in this quadrangle have many resemblances to those in the larger surrounding region. As it is the purpose of this article to consider only the problems surrounding the ore deposits of this region, it will suffice to divide the rocks into two groups—those which are older than the ore deposits and those which are younger.

Rocks Older than the Ore Deposits

Sedimentary Rocks.—The oldest rocks in the quadrangle are siliceous shales and argillites which present uncommonly uniform characters over a large area. Most of the material is dark gray and thin-bedded. As observed casually in some parts of the district, these beds seem to be present in enormous thickness, possibly 30,000 ft. or more, but by careful study of the beds in the area near Bourne (Fig. 1) Pardee[8] was able to conclude that only about 3000 ft. is present. Of this thickness, 600 ft. was represented by a sill of gabbro. The remaining 2400 ft. was separable into seven parts which ranged from thinly laminated black argillite to light gray shale. The material encountered underground presents the same features as that which outcrops, but Pardee concluded that shale made up a larger part of the unit than would be suspected from outcrops. The area east of Sumpter, on both sides of Elkhorn Ridge, contains two belts of limestone outcrops which represent the only material in the section that differs from the dominant shale and argillite. These outcrops are clearly distinct detached blocks of limestone that were once parts of several continuous beds.

Sporadic fossils collected from several of the limestone blocks contain *Fusulina* and indicate that a large part of this group of rocks is Pennsylvanian in age. As an unconformity between argillite beds and overlying limy beds was observed in one locality, it seems probable that beds younger than Pennsylvanian may be present in the quadrangle. Fossils collected recently, by Gilluly[9] in the Keating copper district, east of Baker, indicate the Permian age of a succession of flows and tuffs with thin limestones in that district. It is possible that some thinly laminated greenstones exposed in the Banzette mine and west of Greenhorn may belong to this group of rocks. Limestone and shale with interbedded volcanic material, all presumably of Triassic age, are present in the Eagle Creek Range, east of this quadrangle.[10]

Gabbro.—In several parts of the quadrangle intrusive masses of a gabbroic rock occur in the larger bodies of argillite. In most areas the texture of the gabbro is preserved, although the augite is altered to hornblende; elsewhere most of the original minerals are altered and the original

[8] J. T. Pardee: *Op. cit.*, 87–88.
[9] J. Gilluly: Personal communication, 1930.
[10] W. Lindgren: *Op. cit.*, 579–582.

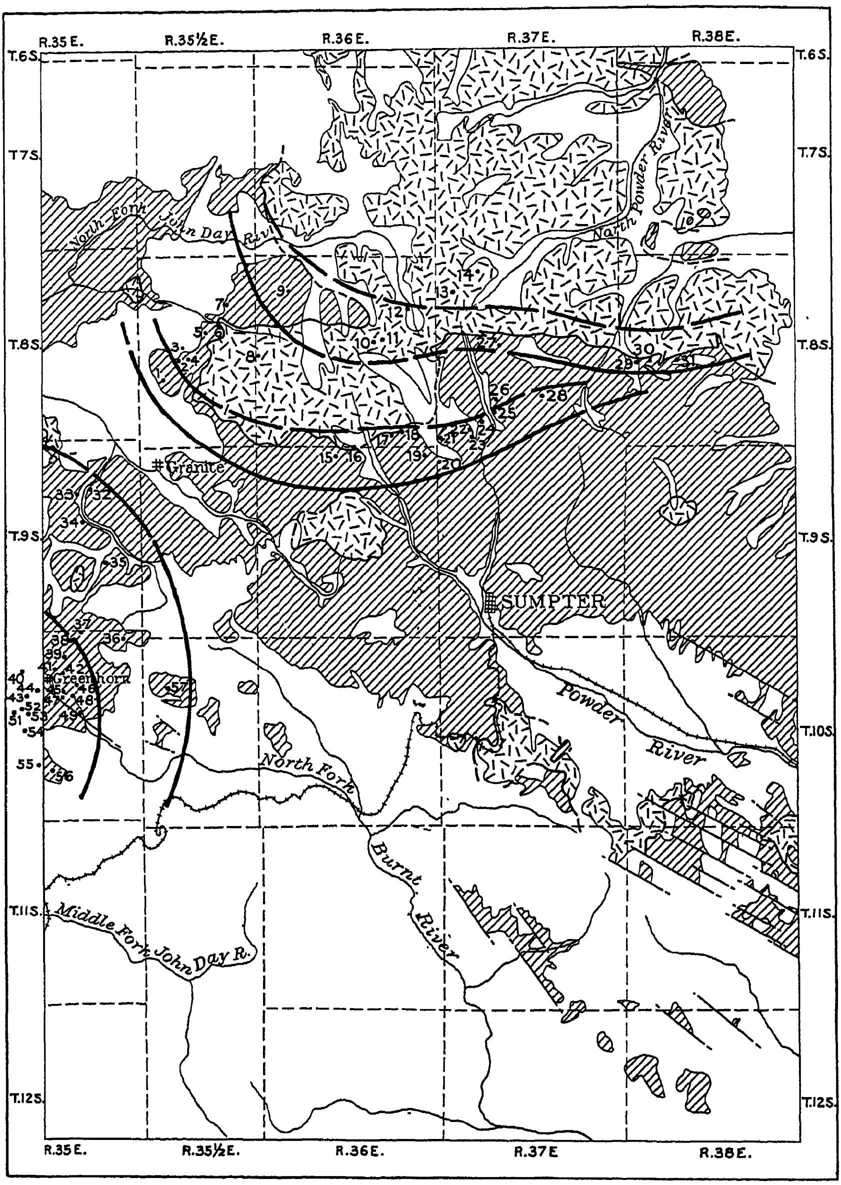

FIG. 1.—GENERALIZED GEOLOGIC MAP OF SUMPTER QUADRANGLE, OREGON. (GEOLOGY BY J. T. PARDEE, D. F. HEWETT, F. J. KATZ AND T. H. ROSENKRANTZ.)

Diagonal lines indicate extent of argillite series and igneous rocks older than the quartz diorite.

Broken lines indicate extent of Mesozoic quartz diorite intrusives.

Clear areas include Tertiary and Quaternary lava flows and sediments.

Black dots indicate positions of mines and numbers agree with list of mines in Table 2.

Heavy lines indicate approximate limits of vein zones described in text.

texture is obliterated. The largest mass is a sill that extends eastward from Bourne across Elkhorn Ridge beyond the limits of the quadrangle, a distance of more than 15 miles. There are other large bodies along the North Fork of the John Day River and on Bulger Flat. Masses of the rock are common near Greenhorn, but many of them are small blocks completely surrounded by serpentine. Both from the relations of the blocks to the serpentine and from the character of their alteration, it is clear that the rock from which the serpentine was derived was intruded after the gabbro.

Many gold-bearing veins in the Greenhorn district lie along the contact of gabbro blocks and serpentine, but elsewhere these rocks are practically free from veins.

Peridotite and Serpentine.—These rocks are not as widespread in the quadrangle as the gabbro, which preceded them, or the quartz diorite, which followed them. Without doubt the serpentine represents an alteration of peridotite or similar rocks. In the region west of Bourne these two rocks tend to occur in the vicinity of bodies of gabbro; viewed broadly, it appears that the more basic rocks were intruded along the same zones of fractures as the earlier gabbro. East of Bourne no peridotite or serpentine has been noted.

Quartz Diorite.—The intrusive granitic rock that underlies about 150 square miles of the northeast corner of the Sumpter quadrangle was termed granodiorite by Lindgren,[11] and to the body he gave the name Bald Mountain batholith. He recognized that it was one of several large bodies of similar rock in northeastern Oregon and that there were also many smaller masses.

The Sumpter quadrangle includes about 854 square miles, or about one-sixth of the area hastily examined by Lindgren, and only a small part of the remainder of the 5000 miles has been reexamined and mapped in detail. As a result of the reexamination of the quadrangle, it may be said that the Bald Mountain batholith is a granitic rock of exceptionally uniform mineral make-up. As names are used at present, quartz diorite rather than granodiorite better conveys the idea of its mineral composition. For the purposes of this inquiry, the exposures may be considered in three groups, as follows:

1. The main mass (Fig. 1) includes the area in the northeast corner of the quadrangle, roughly 9 by 15 miles, limited by the contact with the surrounding sediments on the south and west. Doubtless it extends many miles north and a few miles east under Baker Valley. It will be helpful to recognize the border zone, 2 or 3 miles wide, and the inner portion.

2. The southwestern or Monumental salient, about 4 by 8 miles, which is continuous with the main mass toward the northeast.

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[11] W. Lindgren: *Op. cit.*, 586.

3. The smaller or satellitic bodies, of which 10 are mapped in this quadrangle. The mineral make-up of six of these has been studied in detail.

Although it does not appear in this quadrangle, another large body of this rock occurs west of Greenhorn. The Ben Harrison mine, 4 miles west of the border, lies in it.

It is proposed here that the mineral composition of the many veins that have been worked in the region in such that they may be divided into several groups, which are disposed in broad circular zones around the centers of the two larger quartz diorite intrusions. One center seems to lie near the North Powder Lakes in the main mass of quartz diorite, and the other west of Greenhorn, near the Ben Harrison mine. These zones are indicated on Fig. 1. Mineral-bearing veins appear to be lacking near the minor or satellitic bodies of quartz diorite.

Variations in Composition of Quartz Diorite.—The larger part of the inner portion of the main mass is a granular rock in which the principal feldspar is a sodic plagioclase (andesine); orthoclase is present only here and there. Quartz makes up 10 to 20 per cent. of the rock; biotite and hornblende, 5 to 15 per cent. Biotite is several times as abundant as hornblende. Titanite, magnetite and apatite are minor accessories. In the border zone quartz is present in similar degree, but by contrast hornblende exceeds biotite. The larger part of the Monumental salient closely resembles the inner portion of the main mass but its border also tends to show more hornblende than biotite. The analysis of a sample collected by Lindgren at the head of Lake Creek probably represents the average of the salient as well as the main mass. It contained silica, 71.23 per cent.; alumina, 14.61; ferric oxide, 0.93; ferrous oxide, 1.66; magnesia, 1.01, lime, 3.29; soda, 4.00; potash, 1.92.[12] The mineral composition of the quartz diorite in the smaller intrusive masses varies slightly from place to place but broadly resembles that of the border zone in that hornblende equals or exceeds biotite.

Contact of Quartz Diorite and Sedimentary Rocks.—Tracing the contact of the intrusive westward from the vicinity of the Baisley Elkhorn mine shows that for 12 miles it is a rather simple surface that dips steeply south. Then, as it turns south around the east end of the Monumental salient, the contact surface dips east at a low angle, but farther south and west it again dips steeply south and southwest. North of the Monumental mine it dips steeply north, and north of La Bellevue mine it dips steeply southwest. Except for a distance of several miles, north of Bourne, the contact surface cuts across the bedding of the adjacent sediments. It seems quite clear that both the main mass and the Monumental salient of quartz diorite enlarge downward for some distance below their surface exposures.

[12] W. Lindgren: *Op. cit.*, 587.

The contact surfaces of the smaller masses of quartz diorite dip uniformly outward, so that, in a broad way, they are rather simple dome-shaped bodies.

For most of the belt within which the quartz diorite and sediments are in contact, there is no profound alteration of the sediments. In the wedge that separates the Monumental salient from the main mass, in which La Bellevue mine is located, the sediments are schistose and much altered. The detailed character of this alteration will be described in the final report on the quadrangle.

Dikes.—The petrographic character of the dikes of the quadrangle has not yet been carefully studied, but a preliminary review, taken with the field observations, indicates several conclusions. Most of the pre-mineral dikes appear to fall readily into three lithologic groups that tend to be restricted to definite areas. One of these groups includes pegmatite dikes, which are most abundant in the inner portion of the main mass of quartz diorite. A few of these are 30 to 40 ft. thick and 125 ft. long, but most are much smaller. Most of these dikes show pink feldspar (orthoclase) and quartz, which locally line open cavities; some contain only white feldspar (plagioclase) and quartz.

The dikes of the second group are light colored and fine grained and for present purposes are called aplites. They are found throughout the quartz diorite intrusive.

The dikes of the third group range from fine-grained to porphyritic varieties of quartz diorite to quartz monzonite. Most of these contain more hornblende and biotite than the quartz diorite. They are largely distributed in the zone of sediments, several miles wide, surrounding the main intrusive mass.

It is an interesting feature that most of the dikes in the broad belt that extends eastward from Granite through Bourne and Elkhorn Ridge to the east edge of the quadrangle trend northeast and are therefore nearly parallel to most of the veins of the region. This parallel relation seems to affiliate the vein deposition and the intrusion of the quartz diorite dikes, which represent the final epoch of pre-Tertiary igneous activity in this region. The fractures that are filled by both veins and dikes were probably formed after the larger intrusive masses of quartz diorite were in place and solidified.

Dikes have been observed in many of the mines, but beyond the fact that they are very fine grained and light colored little is known of their original mineral composition, as they are now much decomposed. Many such premineral dikes are shown in the walls of the Independence and Cougar mines, and they are also present in the North Pole, Columbia, Golconda, Ibex and Red Boy mines. In several places it seems clear that some dikes were intruded before the vein-bearing fractures were

formed; afterward other dikes were intruded on the fractures but before the quartz, gold and associated minerals were deposited.

Rocks Younger than the Ore Deposits

In an earlier report[13] the character and relations of the Tertiary igneous rocks have been described. It will be sufficient to state here that neither the faulting nor the process of alteration, which accompanied and followed their extrusion and intrusion, appears to have greatly affected the ore deposits. It will be shown later that the veins record numerous fracturings, but all of these seem to be related to the general process of vein deposition. Relatively few postmineral faults are recorded in the mine workings; they seem to be restricted to the region south of the Blue Mountains, beyond the limits within which most of the veins are known.

Lodes of the Sumpter Quadrangle

Areal Distribution

The locations of the principal veins of the quadrangle, as well as several outstanding veins in the Greenhorn district that lie just west of the border, are shown in Fig. 1. This map shows that although the known veins tend to lie in a broad belt that trends east across the quadrangle, with a cluster of veins near Greenhorn, within this belt they are sporadically distributed. Considered more closely, most of them form groups near seven centers—Greenhorn, Alamo, Granite Creek, McCully's Fork, Cable Cove, Bourne and upper Rock Creek.

If to this picture of groups of veins is added the structural element of their trend and dip, it will appear that most of the veins, except those near Greenhorn and Alamo, form parts of persistent and parallel fracture systems. Thus one can consider that the Cougar, Independence, Magnolia, Buffalo and La Bellevue veins north of Granite occupy a group of nearly continuous and parallel fractures about six miles long. Similarly, the Ibex, Bald Mountain and Belle of Baker-Mammoth veins form a distinct group nearly four miles long. The Highland, Maxwell and Baisley-Elkhorn form another such group. It is not uncommon among residents of the region to assume that mineralized fractures persist from one vein to another a mile or more distant on the strike. Certainly one outstanding example is afforded by the Golconda, Columbia, Eureka-Excelsior and North Pole mines, which have explored one vein for a distance of about 12,000 ft. So far as the writer can learn, however, it remains to be proved that any other vein is continuous for more than 5000 feet.

It may be helpful to state here that although most of the veins of the east-west belt fall readily into a few structural groups, within each group

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[13] J. T. Pardee and D. F. Hewett: *Op. cit.,* 39–47.

there is a progressive change in mineral make-up from the northeast end, near the main intrusive center, outward into the surrounding sediments. Thus, each fracture system tends to exhibit representatives of each of three groups of veins with distinctive mineral make-up. Furthermore, these veins with similar mineral make-up, though belonging to separate structural systems, may be regarded as parts of several zones that lie nearly parallel to the general intrusive contact. Any summary of the geographic distribution of the lodes is a crude expression of several basic geologic relationships.

It is further believed that the veins of the Greenhorn and near-by districts (Red Boy and Bonanza) present a similar zonal distribution, although, on account of greatly differing local geology, the structural relations and mineral make-up of the veins are unlike those of the other belt.

It is believed that the grasp and further understanding of these relationships may be a real aid to the practical problem of exploiting the mines.

Geologic Distribution

In the principal belt most of the veins lie in the sediments; some lie either on or across the contact with the quartz diorite, and a few lie wholly in the intrusive rock, a mile or more from the contact. In this belt the nature of the enclosing rock seems to have had little influence on the attitude, continuity or mineral make-up of the veins. All the veins consist largely of quartz with varying percentages of sulfide minerals; minor calcite or dolomite is present in a few veins (Independence, North Pole).

In the Greenhorn district all the rocks of the region are present, but most of the veins are on the contact of gabbro masses with the surrounding serpentine. Compared with those of the principal belt, they are much more numerous, lie in diverse attitudes, and are much less persistent. Veins of quartz are known in serpentine, gabbro and argillite, but a larger proportion of the total have carbonate gangue (dolomite.)

Major Features of the Veins

Explored Length and Depth.—Table 1 summarizes the length and vertical depth below the outcrop to which the principal veins of the region have been explored. It emphasizes the extraordinary persistence of the vein locally known as the "Mother Lode," worked by the Columbia, Eureka-Excelsior and North Pole mines. If the vein worked in the Golconda mine represents the southwestern extension of the "Mother Lode," as seems to be the case, the explored length is nearly 15,000 ft. The explored depth on this vein, 2500 ft., is the distance from the outcrop on the northeast end of the North Pole claims to the bottom of the Columbia shaft, 918 ft. deep.

TABLE 1.—*Explored Length and Depth of the Principal Veins*

Mine	Length, Ft.	Depth, Ft.	Mine	Length, Ft.	Depth, Ft.
Cougar	1,600	350	Mother Lode:		
Independence	1,050	300	Columbia		
Magnolia	950	280	Eureka-Excelsior	12,000	2,500
Buffalo	800	400	North Pole		
La Bellevue	2,200	500	Buckeye	2,500?	900
Monumental	500	700	Highland	2,800	600
Imperial Eagle	2,300	700	Maxwell	2,500?	1,200
Ibex	2,950	600	Baisley-Elkhorn	1,500	950
Belle of Baker	700	400	Bonanza	750	1,250
			Red Boy	1,200	500
			Ben Harrison	750	560

Without doubt, the veins in these mines extend much farther, both horizontally and vertically, than the distances stated above.

Strike and Dip.—It has already been noted that there is a surprising uniformity of strike and dip of the veins that surround the main mass of quartz diorite; with only a few exceptions, they trend northeast and dip steeply southeast. Notable exceptions to the prevailing southeast dip include La Bellevue, Eagle, Highland and Baisley-Elkhorn. In an early study of the district, Pardee[14] concluded that the Mother Lode lies on a fault in which the horizontal displacement (1800 ft.) greatly exceeds the vertical. This conclusion fortifies the impression gained from other mines in this belt that the veins lie along minor northeasterly faults on which most of the movement has been horizontal. On the other hand, the trend and dip of the veins in the Greenhorn area are diverse. A possible explanation may lie in the character of the rocks that prevail in that part of the region. Most of the veins lie in serpentine or gabbro.

Structure of the Veins.—It will serve a purpose to distinguish between veins that are simple in structural make-up, on the one hand, and those that are composite, on the other. By the term "simple" as applied to veins is meant those that, so far as close examination can show, are made up of a single strand of gangue and sulfides introduced during one epoch of mineralization. In this region simple veins are single strands of quartz or carbonate minerals which either fill the entire space between the walls or cement angular fragments of wall rock but which on close examination either underground or in polished specimens show no evidence of brecciation of quartz or introduction of more quartz in adjacent or near-by strands. On the other hand, composite veins are those which are made up of many distinct strands of quartz or carbonate or those which reveal evidence of repeated brecciation and cementation by quartz

[14] J. T. Pardee: *Op. cit.*, 91–92.

or carbonate. It is here interpreted that each strand or successive brecciation and cementation records a new epoch or pulsation of mineralization, even though, in a geologic sense, the whole period of vein formation was brief.

In this region simple veins with carbonate gangue are common in the Greenhorn district; for instance, the Golden Eagle vein is a single strand of dolomite with sparse sulfide minerals, which does not appear to have been broken. Simple veins of quartz with or without sulfides are very common throughout the region, but few, if any, have been mined singly as a source of gold and silver. Some simple veins of gold-bearing quartz 1 in. or more wide have been mined in the Bonanza mine, and doubtless in other places.

It is an impressive feature of the region that almost all the veins, and especially those which have been the source of most of the production, are composite; they are the result of several epochs of mineralization. As recorded below, the general order of deposition of the minerals is as follows: (1) Quartz in several epochs of deposition, the first generally replacing the country rock; (2) sulfides, sulfarsenides, and sulfantimonides; (3) gold. In most of the veins examined that contain quartz, sulfides and gold, the relations of these minerals reveal at least three epochs of mineralization, each distinguished by an uncommonly high proportion of that mineral or minerals.

Lindgren recognized this characteristic of the veins when he wrote:[15] "Secondary changes are frequent in the veins. In many places the quartz shows undulous extinction and is traversed by crushed zones. Whole masses of primary quartz with arsenopyrite in concentric deposition are crushed and recemented by quartz."

Lindgren also said:[16] "The quartz here is full of small curved and concentric streaks of arsenopyrite, in general not parallel to the walls and frequently sharply cut off. The detailed examination shows that this irregularly concentric structure is due to crustification but that by subsequent motions the crustified mass has been broken and recemented, in places forming a real breccia."

Fig. 2 shows the composite character of the "Mother Lode" explored in stope above the No. 3 tunnel of the North Pole mine. It is interpreted as showing at least three and possibly four epochs of vein deposition, separated by movements roughly parallel to the walls. During the first epoch, the parts of the vein marked 2 and 3 were deposited; during the second, 4 and 5; finally, 7. Of the three strands of the vein, only 7 is ore of good grade; it is reported by the foreman to contain gold to the

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[15] W. Lindgren: *Op. cit.*, 620.

[16] W. Lindgren: *Op. cit.*, 660. He, doubtless, refers to such structures as that shown in Fig. 10.

extent of $100 to $150 per ton. The detailed character of the part of the vein marked 7 is shown in Figs. 13 and 14, which are discussed on page 29.

The composite character of many veins in this region is revealed, however, only by close examination of polished and etched specimens and of thin sections. Even though a vein may be made up of several distinct strands, close examination commonly reveals that several strands represent quartz veins crushed and recemented by quartz several successive

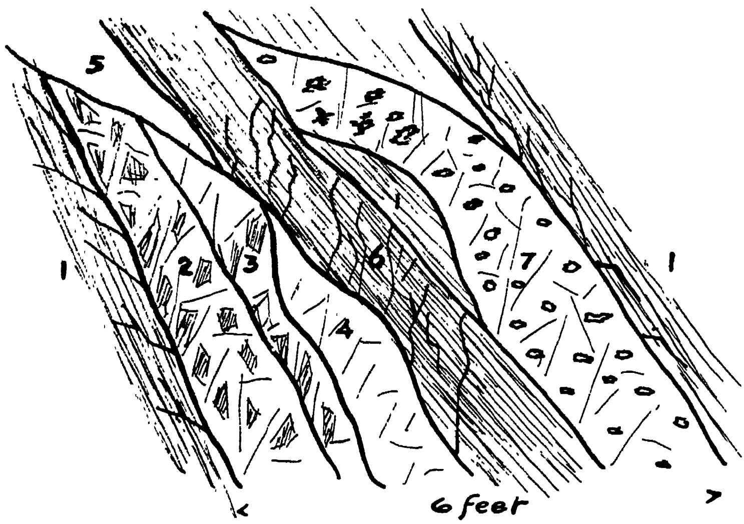

FIG. 2.—NORTH POLE VEIN. STOPE 30 FT. ABOVE NO. 3 TUNNEL, 1800 FT. FROM MOUTH.

1. Sheared carbonaceous argillite, few quartz stringers.
2. Argillite breccia, largely replaced by quartz.
3. Argillite breccia, partly replaced by quartz.
4. Argillite breccia, completely replaced by quartz.
5. Same as 4.
6. Dark argillite, few quartz veins.
7. Clean quartz containing knots of arsenopyrite and pyrite. Source of specimen is shown in Figs. 13 and 14.

times. The Ibex vein is of this type. It has been explored underground for 2950 ft. and the thickness between the outer walls ranges from 3 to 16 ft. For most of the distance it contains three or four strands of quartz, and some of these reveal three distinct epochs of deposition of quartz (Figs. 8 and 9). In this vein the principal sulfide mineral is tetrahedrite, which was deposited in a late stage of the formation of the vein. The other sulfides present, pyrargyrite, argentite, and cinnabar, are probably supergene. The cinnabar has clearly been formed by the decomposition under weathering of the mercurial variety of tetrahedrite, schwatzite.

Detailed Features of the Veins

Mineralogy and Texture.—No attempt will be made in this paper to record the entire list and properties of the minerals observed in the veins of this region and their paragenetic relations, but the outstanding features of the minerals and their relations as revealed by the study of many polished and thin sections have direct bearing on the hypothesis presented here. It seems best to assemble this record by groups of mines according to the zones in which they lie. This record does not duplicate the list of minerals recorded in Table 2 but supplements it.

The photographs that accompany this paper have been chosen to present a part of the evidence which tends to show (1) the kind, state of aggregation and amount of the sulfide minerals characteristic of the veins, (2) the general succession of quartz and sulfide minerals, and (3) the extraordinary amount of crushing that the vein minerals have undergone at several epochs.

Ore Deposits Zonally Disposed around Bald Mountain Batholith

La Bellevue Mine, Inner Zone.—La Bellevue mine explores several veins, but by far the greater part of the work follows one vein. Where this vein is narrow it consists largely of sheared chloritic gouge, but where it is 2 to 4 ft. wide it is made up of two or more strands of quartz with sulfides. The quartz contains numerous phantom angular fragments of schist. A typical specimen of ore, sawed and polished, shows that early quartz and coarsely crystalline pyrite have been crushed; then arsenopyrite was deposited; this was followed by further crushing and finally by the deposition of more quartz with minor quantities of blende, chalcopyrite and galena. Studies of the ores made by an engineer who examined the mine recently indicated that the massive pyrite and arsenopyrite rarely carried much gold but that the content was higher when galena, chalcopyrite and antimony minerals were present.

Imperial Eagle Mine, Inner Zone.—The tunnels of the Imperial Eagle mine explore a branching system of veins which contain essentially the same minerals. Each vein contains one or two but rarely more strands of quartz, which attain a maximum width of 3 or 4 ft. The veins lie in decomposed quartz diorite gouge which is limited by walls of fresh diorite, commonly 4 ft. but sometimes as much as 16 ft. apart. A specimen from the Harvey vein (Fig. 3) shows considerable coarse arsenopyrite which has been crushed and recemented by later quartz; after another epoch of crushing, blende and galena were deposited. A thin section of this material (Fig. 4) shows numerous crystals and angular fragments of arsenopyrite in a matrix of interlocking quartz crystals. Later veinlets of calcite partly replace this quartz.

Baisley-Elkhorn Mine, Inner Zone.—The Baisley-Elkhorn mine explores one principal vein, largely in quartz diorite, and several subsidiary

TABLE 2.—*Mines Exploiting Veins Related to Bald Mountain Batholith and Greenhorn Intrusive*

Number on Map	Name	District	Development	Relationships	Minerals	Milling Record	Production
			VEINS RELATED TO BALD MOUNTAIN BATHOLITH				
13	Loy's	North Powder	4 tunnels, longest 500 ft.	Vein in quartz diorite	Quartz, pyrite, chalcopyrite	None	None
14	Flying Cloud	North Powder	1 tunnel, short	Vein in quartz diorite	Quartz, pyrite, chalcopyrite	None	None
			INNER ZONE				
9	La Bellevue	Granite	4 tunnels, 8,500 ft. of work; 500 ft. below outcrop	Vein in quartz-mica schist and gneiss. Strike N. 45° E., dip 63° NW.	Quartz, arsenopyrite, pyrite, blende, galena, chalcopyrite, pyrrhotite, silver sulfides	Not available; estimated ratio of sulfides to quartz, 1:5 or 1:10. Little free gold.	$ 300,000[a]
10	California	Cable Cove	6 tunnels, longest 1,750 ft.	Vein in quartz diorite	Quartz, arsenopyrite, pyrite, blende, galena, chalcopyrite	Not available. Estimated ratio of sulfides to quartz, 1:10	$ 40,000[a]
11	Imperial Eagle	Cable Cove	6 tunnels, longest 1,700 ft.; attains 600 ft. below outcrop	4 veins in quartz diorite, N. 20°–60° E., all dip SE. except Eagle	Quartz, arsenopyrite, pyrite, blende, galena, chalcopyrite, molybdenite	Not available. Estimated ratio of sulfides to quartz, 1:10. Gold 10 per cent. free. Ratio of gold to silver about 1:10	$ 100,000[a]
12	Last Chance	Cable Cove	2 tunnels; longer 450 ft.	Vein in quartz diorite. Strike N. 50° E., dip 80° SE.	Quartz, arsenopyrite, pyrite, blende, galena	Not available. Concentration ratio about 10:1	$ 5,000[a]
27	Mountain View	Bourne	2 tunnels	Vein in argillite	Quartz(?)	Not recorded	$ 90,000[a]

29	Highland	Rock Creek	About 7,000 ft. of work; longest drift 2,000 ft.; deepest work 600 ft. below outcrop	Vein in argillite; strike N. 75° E., dip 85° N.	Quartz, pyrite, blende, galena, arsenopyrite, chalcopyrite, tetrahedrite	Concentration ratio 5:1 to 12:1, average 7:1. Ratio of gold to silver about 1:10	$ 300,000[a]
30	Maxwell	Rock Creek	6 tunnels; longest 900 ft.; aggregate 5,000 ft.	Vein in altered argillite. Strike N. 70° E., dip 70° to 80° SE.	Quartz, pyrite, blende, galena, arsenopyrite, siderite, calcite, fuchsite	Concentration ratio about 10:1	Not available
31	Baisley-Elkhorn	Rock Creek	About 10,000 ft. of work. Principal work is crosscut 2,300 ft. to vein and 1,100 ft. drift on vein zone 950 ft. below outcrop.	Vein in quartz diorite close to contact with argillite. Strike N. 60° E., dip 85° NW.	Quartz, pyrite, blende, chalcopyrite, calcite	Concentration ratio from 5:1 to 7:1. Gold 20 to 25 per cent. free	$ 950,000[a]
	INTERMEDIATE ZONE						
5	Buffalo	Granite	2 tunnels, about 4,000 ft. of tunnels and drift; explored 400 ft. below outcrop	3 veins in argillite near quartz diorite contact. Strike N. 20° E., dip 65° to 80° NW.	Quartz, pyrite, galena, tetrahedrite, dolomite, chalcopyrite	Concentration ratio about 10:1; tends to decrease with depth. Bullion: gold 600, silver 350. Gold 20 per cent. free. Ratio of gold to silver, 1:10	$ 350,000[a]
6	Blue Ribbon	Granite	2 tunnels, about 2,000 ft. of tunnels and drifts, attaining 190 ft. below outcrop	Vein in argillite intruded by basic dikes. Strike N. 65° E., dip 85° SE.	Quartz, pyrite, tetrahedrite	None	Small
7	Standard	Granite	1 tunnel 400 ft.	Vein in quartz-mica schist	Quartz, pyrite, arsenopyrite	None; about 10 per cent. sulfides	None

[a] Estimated.

TABLE 2.—(*Continued*)

Number on Map	Name	District	Development	Relationships	Minerals	Milling Record	Production
8	Monumental	Granite	2 tunnels and shaft, total about 4,000 ft.; attaining 700 ft. below outcrop	Several veins in quartz diorite. Strike N. to N. 20° E., dip 65° NW.	Quartz, arsenopyrite, pyrite, blende, galena, tetrahedrite, pyrargyrite	None recorded. Ratio gold to silver, 1:20 or more	$ 100,000[a]
3	Magnolia	Granite	3 tunnels, lowest and longest 1,000 ft.; attains 280 ft. below outcrop	Vein in argillite	Quartz, pyrite, marcasite, siderite, arsenopyrite	Meager record. Gold 15 to 30 per cent. free	Small
4	Ajax	Granite	2 tunnels, lower about 500 ft.	Vein in argillite	Quartz, pyrite	Meager record. Gold nearly equals silver	$ 40,000[a]
2	Independence	Granite	3 tunnels, total 3,500 ft.; lowest follows vein 950 ft.; attains 300 ft. below outcrop	Vein in argillite (carbonaceous); many premineral dikes. Strike N. 55° E., dip 66° SE	Quartz, dolomite, pyrite, arsenopyrite, pyrargyrite, chalcedony	Meager record. Ratio gangue to sulfides, 20:1. Ratio of gold to silver, about 1:5	Not available

OUTER ZONE

Number on Map	Name	District	Development	Relationships	Minerals	Milling Record	Production
1	Cougar	Granite	3 tunnels, total 3,000 ft.; middle follows vein 1,300 ft.; attains 350 ft. below outcrop	Vein in argillite; many premineral dikes. Strike N. 45° E., dip 78° SE.	Quartz, pyrite	Meager record. Ratio of gangue to sulfides high. Gold nearly equal to silver. Little free gold	Small
15	Ibex	McCully	3 tunnels, total 5,500 ft.; lowest tunnel follows vein 2,700 ft.; attains 600 ft. below outcrop	Vein in argillite near quartz diorite, few premineral dikes. Strike N. 25° E., dip 60° to 80° SE.	Quartz, pyrite, tetrahedrite, pyrargyrite, calcite, secondary cinnabar	Meager record. Ratio of gangue to sulfides low (less than 5 per cent. ?) Ratio of gold to silver about 1:10 or less. Gold 30 per cent. free	Small

16	Bald Mountain	McCully	Shaft and drifts with tunnel 300 ft.; total about 3,000 ft.	Vein in argillite (extension of Ibex?) Strike N. 60° E., dip 70° SE.	Quartz, pyrite	Meager record. Ore similar to Ibex. Gold 30 per cent. free	Small
17	Belle of Baker	McCully	Shaft 385 ft.; with 2,000 ft. of drifts	Vein on contact of argillite and diorite (extension of Mammoth)	Quartz, pyrite, roscoelite, calcite. Free gold	Meager record Ratio of gangue to sulfides very low. Gold largely free	$ 400,000[a]
18	Mammoth	McCully	Shaft 300 ft. with drifts	Vein on contact of argillite and diorite.	Quartz, pyrite	Meager record. Bullion 500 to 600 fine	$ 40,000[a]
19	Annalulu	Silver Creek	Tunnels	Vein in argillite	Quartz	No record	No record
21	Mountain Belle	Silver Creek	Shaft 300 ft. and tunnel	Vein in argillite	Quartz, pyrite	No record	No record
20	Mayflower	Silver Creek	2 tunnels, lower 510 ft. crosscut, and drifts	2 veins in argillite. Strike N. 75° E., dip 75° SE.	Quartz, pyrite, chalcopyrite, tetrahedrite, roscoelite, fuchsite	No record	None
22	Climax	Bourne	2 tunnels, lower 550 ft. crosscut, and drifts	3 veins in argillite. Strike N. 65° E., dip 60° NW.	Quartz, fuchsite, sparse sulfides	No record	Small
23	Golconda	Bourne	Shaft 510 ft.; total work about 7,000 ft.	3 veins in argillite. Strike NE., dip NW.	Quartz, pyrite, arsenopyrite, marcasite, possibly other sulfides, much fuchsite	Meager record. Ratio of concentration 7:1 to 15:1. Free gold 40 to 50 per cent. Ratio of gold to silver in bullion 720 to 220	$ 600,000[a]
24	Columbia	Bourne	Shaft 918 ft.; total work about 50,000 ft.	Vein in argillite, Strike NE., dip SE.	Quartz, pyrite, stibnite, fuchsite, roscoelite	Record not available. Ratio of concentration 10:1 to 15:1. Free gold 40 per cent.	$4,000,000[a]

[a] Estimated.

TABLE 2.—(*Continued*)

Number on Map	Name	District	Development	Relationships	Minerals	Milling Record	Production
25	Eureka-Excelsior	Bourne	Shaft 702 ft.; total work about 20,000 ft.	Vein in argillite; strike NE.	Quartz, pyrite, arsenopyrite	Ratio of concentration, gangue to sulfides 25:1 to 20:1. Free gold less than 5 per cent. Ratio of gold to silver about 1:2	$1,750,000[a]
26	North Pole	Bourne	5 tunnels, total work about 13,000 ft.; attains 1,100 ft. below outcrop	Vein in argillite, strike NE., dip SE.	Quartz, pyrite, arsenopyrite, stibnite, tetrahedrite, fuchsite, roscoelite, hessite	Free gold, 6 to 20 per cent.	$2,000,000[a]
28	Buckeye	Bourne	4 tunnels, total work about 4,000 ft.; attains 900 ft. below outcrop	2 veins in argillite. Strike N. 60° E., dip SE.; strike N. 85° E., dip S.	Quartz with meager sulfides	Not recorded	$ 6,000[a]
	VEINS RELATED TO GREENHORN INTRUSIVE						
50	Ben Harrison[b]	Olive Lake	2 tunnels and shaft, about 4,000 ft. work; attains 600 ft. below outcrop	Vein in diorite, strike NE., dip SE.	Quartz, stibnite, tetrahedrite, pyrite	Meager record. Ratio of gangue to sulfides, about 25:1. Ratio of gold to silver, about 1:50	$ 350,000[a]
51	Banner	Greenhorn	Shaft 200 ft.; with 500 ft. of drifts; not accessible	Vein in serpentine, probably near gabbro. Strike W., dip S.	Quartz, dolomite	None recorded	Not recorded

52	Diadem	Greenhorn	Not accessible	Vein in serpentine	Dolomite, galena, chalcopyrite	None recorded	Not recorded
53	Snow Creek	Greenhorn	Tunnel 1,400 ft. to shaft 250 ft. with levels	Vein in serpentine. Strike W., dip S.	Quartz, chalcopyrite, galena	Not available	Not recorded
54	Eccentric	Greenhorn	Open cut and 2 tunnels, 250 ft.	Vein on contact of serpentine and gabbro	Dolomite, galena, chalcopyrite	Not available	Small
55	Gold bullion	Greenhorn	Shaft 60 ft.	Vein in sheared gabbro	Quartz, pyrite, chalcopyrite	None recorded	Not recorded
56	Listen Lake	Greenhorn	Shaft 120 ft.	Vein in sheared gabbro	Quartz, pyrite, chalcopyrite	None recorded	Not recorded
40	Muscatine	Greenhorn	Crosscut tunnel 1,000 ft.; drift on vein 120 ft. to shaft 200 ft. deep	Vein on contact of serpentine and gabbro. Strike N., dip W.	Dolomite, pyrite	None recorded	Not recorded
43	Banzette	Greenhorn	Tunnel 1,600 ft. to shaft 100 ft. deep	Vein in sheared greenstone. Strike generally E., dip N.	Quartz, galena	None recorded	Small
44	West Side	Greenhorn	Shaft, not accessible. Tunnel 500 ft.; accessible	Vein in serpentine (?)	Dolomite, quartz, chalcopyrite, galena, opal	None recorded	Small
45	Dodo	Greenhorn	2 shafts 80 and 100 ft.; not accessible	Vein in sheared serpentine	Quartz, chalcopyrite	Not recorded	Not recorded
46	Rabbit	Greenhorn	1 shaft 67 ft., drifts	Vein in metagabbro. Strike N. 5° E., dip 70° W.	Quartz	Not recorded	Small

[a] Estimated.
[b] Not shown on Fig. 1.

TABLE 2.—(*Continued*)

Number on Map	Name	District	Development	Relationships	Minerals	Milling Record	Production
47	Phoenix	Greenhorn	3 tunnels about 1,500 ft.	Vein in serpentine	Quartz, chalcopyrite	None recorded	Small
48	Don Juan	Greenhorn	Tunnel, not accessible	Vein in serpentine	Dolomite, chalcopyrite, pyrite	None recorded	Small
49	Golden Eagle	Greenhorn	3 tunnels, about 2,000 ft. Attain depth of 175 ft.	Veins in serpentine	Dolomite, galena, chalcopyrite	None recorded	$ 75,000[a]
41	I. X. L.	Greenhorn					
42	Humboldt	Greenhorn	Shaft 60 ft.; with drifts, total 400 ft.	Vein in gabbro	Quartz	None recorded	Small
39	Golden Gate	Greenhorn	1 tunnel 700 ft.	Vein in argillite	Quartz	None recorded	Small
38	Belcher	Greenhorn	2 tunnels about 3,500 ft. Attains 350 ft. below outcrop	Vein in argillite near andesite contact	Quartz, pyrite	None recorded	Small
37	Royal White	Greenhorn	Tunnel 600 ft.	Vein in argillite. Strike NE., dip NW.	Quartz	None recorded	Small
34	Pyx	Greenhorn	Several tunnels and shaft, total 500 ft.	Vein in argillite	Quartz, pyrite	None recorded	Small
35	Maid of Oregon	Alamo	3 tunnels about 1,500 ft.	2 veins in argillite	Quartz, marcasite	None recorded	Not recorded
36	Black Jack	Alamo	1 tunnel 2,000 ft.; attains 700 ft. below outcrop	Vein in argillite	Quartz	None	None

33	Red Boy......	Alamo	3 tunnels and 300 ft. shaft, total about 5,000 ft.; attains 500 ft. below outcrop	3 veins in argillite, cut by several premineral dikes	Quartz, pyrite, arsenopyrite, chalcopyrite	Meager record. Ratio of concentration, 25:1. Free gold 75 to 85 per cent., bullion 520:450	$1,000,000[a]
32	Blue Bird....	Alamo	1 tunnel 3,500 ft.	Vein in argillite	Quartz, pyrite, arsenopyrite	None recorded	Small
57	Bonanza...	Geiser	3 tunnels, shaft 1,200 ft. deep; total about 18,000 ft.	Vein in argillite and greenstone near serpentine contact. Strike N. 55° W., dip 80° SW.	Quartz, calcite, pyrite, arsenopyrite	Meager record. Ratio of gangue of sulfides, 20:1 to 25:1. Free gold, 70 per cent. Fineness of bullion 750:240	$1,000,000[a]

[a] Estimated.

parallel veins. As exposed on the lowest tunnel, 950 ft. below the outcrop, the vein contains one or two strands of quartz 6 to 12 in. wide, containing coarse sulfides in the midst of quartz diorite gouge. A polished specimen reveals the following order of deposition: coarse pyrite, quartz, blende and galena. No distinct epoch of crushing is shown.

Buffalo Mine, Intermediate Zone.—The three parallel veins explored in the Buffalo mine have general resemblances. There is generally a

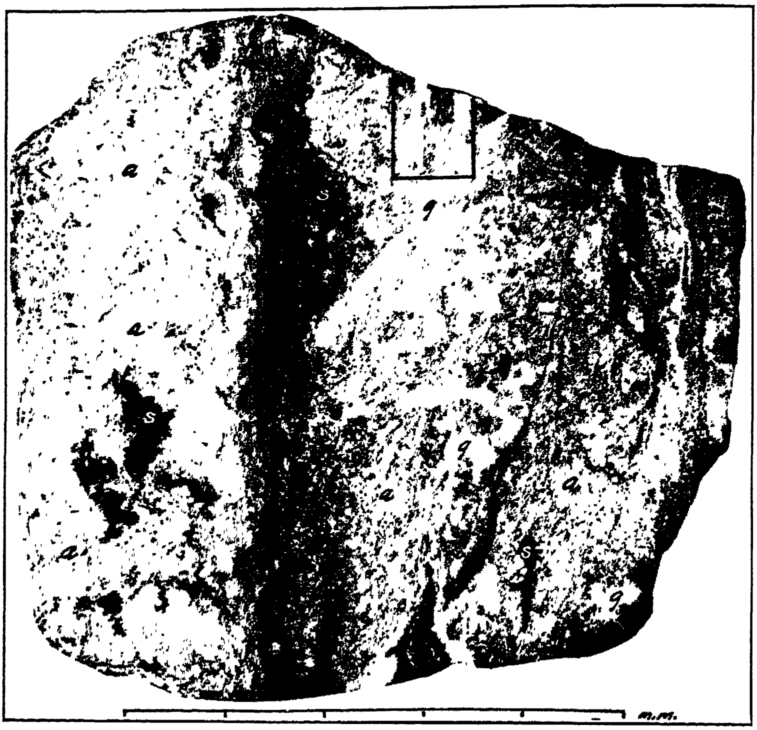

FIG. 3.—SPECIMEN FROM HARVEY VEIN, IMPERIAL EAGLE MINE, CHARACTERISTIC OF INNER VEIN ZONE.

Largely arsenopyrite *a* containing a little quartz that has been crushed and recemented (1) by quartz *q* and (2) by sphalerite *s* and galena *g*. Position of thin section shown in Fig. 4 is indicated at top.

persistent footwall strand of quartz 6 to 15 in. wide, with coarse and fine sulfides, and less persistently a hanging-wall strand. The examination of a suite of polished specimens from veins 2 and 3 indicates that dolomite, possibly with some quartz, was the earliest mineral (Fig. 5); this was crushed, and quartz was deposited, followed by coarse pyrite and arsenopyrite. After further crushing, sphalerite, then chalcopyrite, tetrahedrite and galena were deposited. Thin sections indicate that after the early crushing some quartz fragments were secondarily enlarged; in other places long blades of quartz seem to have grown in the breccia (Figs. 6

and 7) but there has not been widespread recrystallization. Pyrite and arsenopyrite seem to have been deposited largely by replacing quartz.

Monumental Mine, Intermediate Zone.—The workings of the Monumental mine have encountered not less than 12 veins, but most of the work has been done on 4. All of these are rather simple single strands of quartz, largely from 2 to 15 in. wide, and although other sulfides were common in the upper workings, arsenopyrite is the most abundant on the lower levels, about 700 ft. below the outcrop. An examination of polished and thin sections shows that an early simple quartz vein was

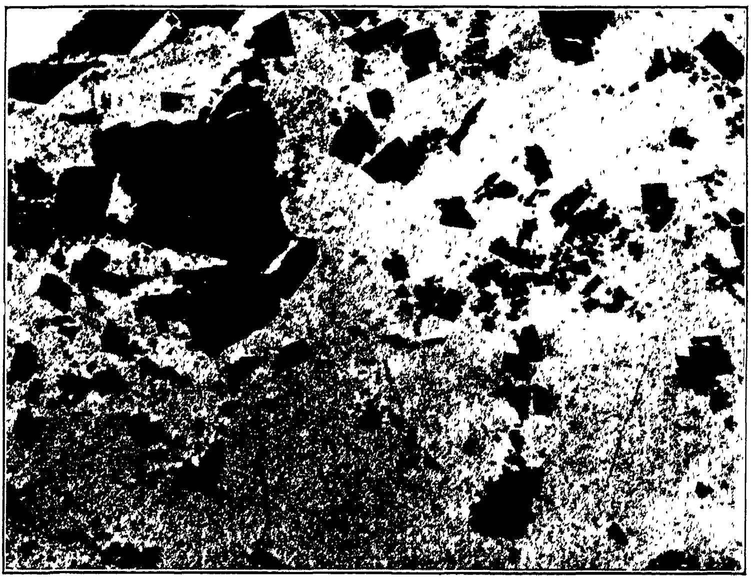

FIG. 4.—SPECIMEN FROM HARVEY VEIN, IMPERIAL EAGLE MINE. × 30. Shows many angular fragments of arsenopyrite embedded in granular quartz.

finely crushed and recemented by pyrite, arsenopyrite and quartz. Later, calcite veins were deposited, in part by replacing quartz.

Independence Mine, Intermediate Zone.—The Independence mine explores a single vein of composite type. The width of the vein commonly ranges from 3 to 4 ft. but locally attains 5 or 6 ft. between a persistent hanging wall and a less definite footwall. It generally contains a single strand of quartzose ore but locally shows four strands. Structurally each strand has a complicated make-up. The simplest parts of the vein show alternate layers of quartz containing minute arsenopyrite crystals and dolomite that surround angular fragments of argillite. Elsewhere corroded angular fragments of dolomite are enveloped and partly replaced by quartz containing minute grains of pyrite, arsenopyrite and blende.

Cougar Mine, Outer Zone.—The Cougar vein consists largely of decomposed argillite and premineral dike rock, and the meager quartz and sulfides are confined to poorly defined simple lenses. There has been

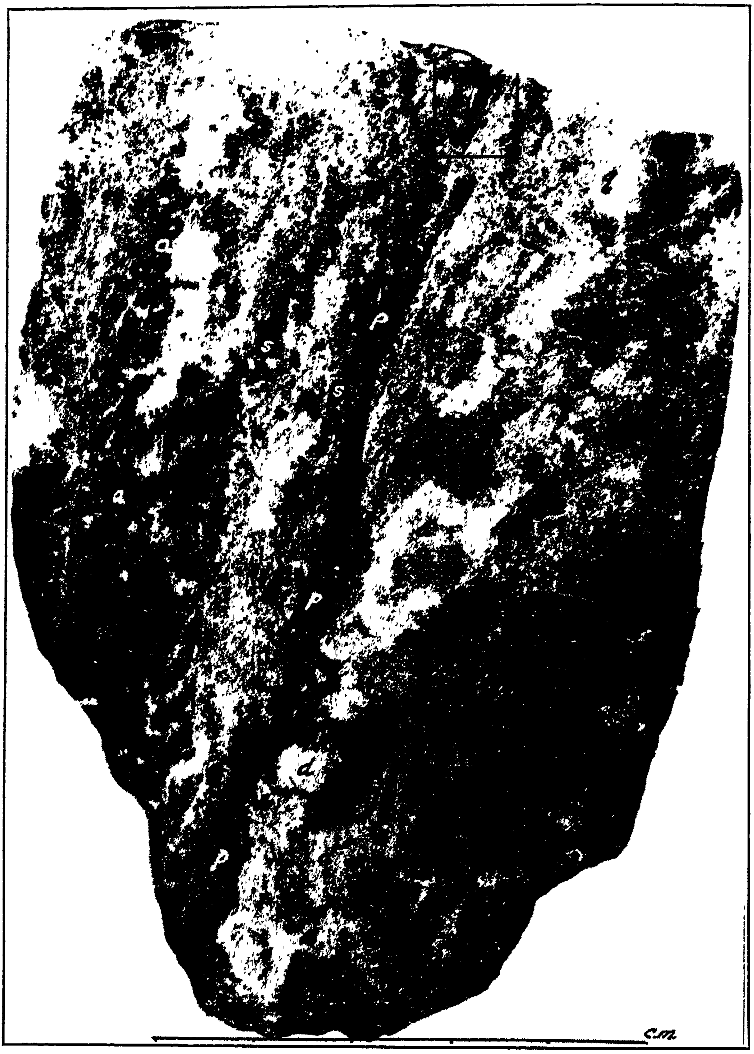

FIG. 5.—SPECIMEN FROM BUFFALO VEIN NO. 2; INTERMEDIATE VEIN ZONE.
Shows sporadic fine-grained sulfides in quartz breccia *q* which cements an earlier breccia of dolomite *d*. Arsenopyrite grains *a*, pyrite *p*, sphalerite *s*. Position of thin sections shown in Figs. 6 and 7 is indicated at top.

only slight replacement of argillite by quartz, and the early white quartz of many veins in the district is lacking here. A specimen of unaltered vein from the face of the middle tunnel, 300 ft. below the outcrop, shows

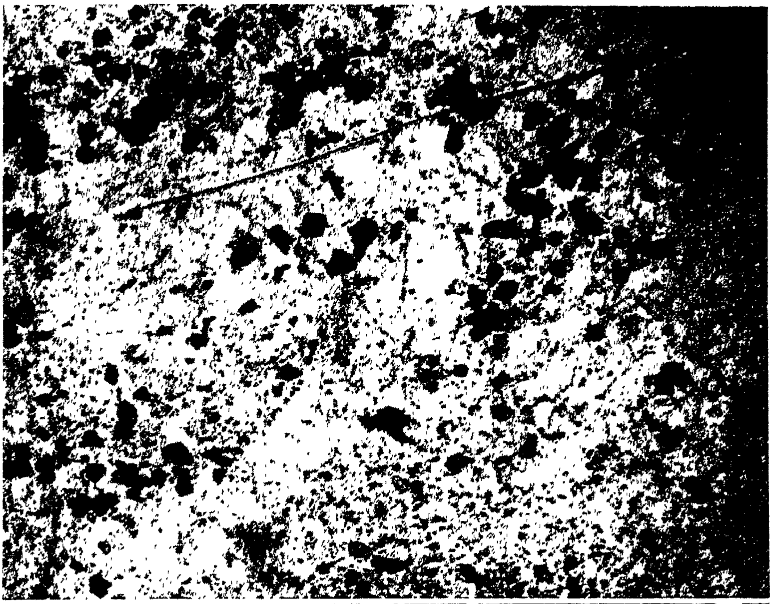

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FIG. 6.—THIN SECTION FROM BUFFALO VEIN NO. 2. × 30.
Shows pyrite crystals developed in quartz breccia, cemented by secondary quartz.

FIG. 7.—THIN SECTION SAME AS FIG. 6, WITH CROSSED NICOLS. × 30.
Shows secondary blade of quartz developed in early quartz breccia.

angular fragments of carbonaceous argillite enveloped first in quartz then in quartz and pyrite. No other sulfide was noted.

Ibex Mine, Outer Zone.—The Ibex mine explores a vein that is generally made up of several strands of quartz, each of which records a com-

Fig. 8.—Specimen from Ibex vein; outer vein zone.

Showing early white quartz containing phantoms of argillite fragments $q1$, broken and cemented by sulfide-bearing quartz veins $q2$, broken and cemented by quartz $q3$. Position of thin section shown in Fig. 9 is indicated by circle near upper right corner.

plicated history. For much of its explored length (2950 ft.) the width of the vein ranges from 3 to 7 ft., and locally the distance between walls attains 12 and even 16 ft., but only rarely does the quartz exceed 5 ft. The remainder is sheared argillite or gouge. Each of the strands of quartz shows numerous angular fragments of argillite, most of which are largely replaced by quartz. Generally, however, only one or two show

sulfides. Numerous specimens from these strands record three epochs of quartz deposition. Thus in Fig. 8 the first quartz is white and coarse and encloses partly replaced argillite fragments; the second quartz was deposited as horizontal crusted veinlets which contain sulfides (largely tetrahedrite); the latest quartz cements a breccia of the earlier material along vertical fractures. Thin sections amply confirm this interpretation but do not reveal evidence of secondary growth of the grains or recrystallization (Fig. 9).

Belle of Baker and Mammoth Mines, Outer Zone.—Although the workings of the Belle of Baker and Mammoth mines have not been

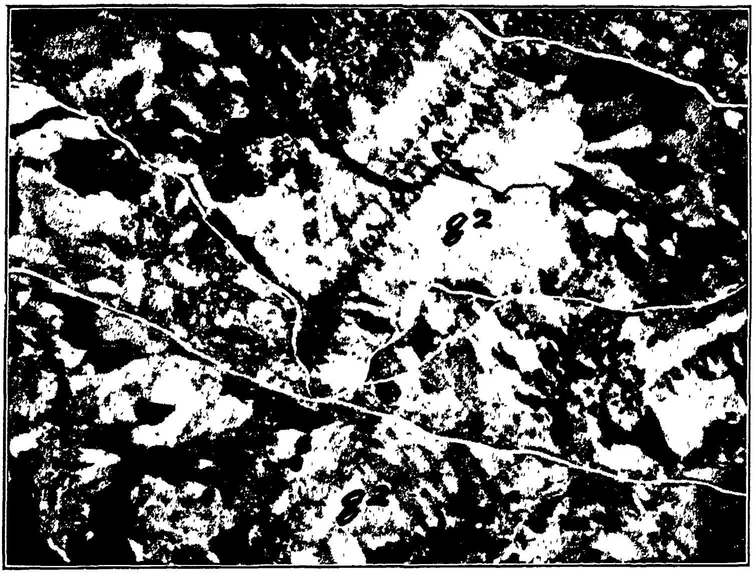


FIG. 9.—THIN SECTION FROM IBEX VEIN. × 30.
Showing fragments of quartz veinlets *q*2 displaced and cemented by later quartz.

connected, both mines appear to exploit a single vein. It is a composite vein made up of as many as five strands of quartz-cemented argillite breccia separated by strands of gouge. From place to place one of these strands contains narrow veinlike bodies that show small spherical masses of roscoelite with which native gold is associated. Examination of a polished section of such material reveals three generations of quartz, each later one cementing a breccia of the earlier. The roscoelite appears to be associated with the quartz of the middle generation. Milling operations show that the content of sulfides, pyrite and arsenopyrite is small.

Mayflower Mine, Outer Zone.—The Mayflower mine explores two parallel veins, each a single strand of quartz but composite in make-up. The southwest or widest vein is largely quartz which replaces and cements

angular fragments of argillite breccia. Quartz of three epochs appears to be discernable and the sulfide minerals which appear in small amount are the latest minerals. The order of deposition is quartz, pyrite, chalcopyrite and tetrahedrite.

"Mother Lode": North Pole, Eureka-Excelsior and Columbia Mines, Intermediate and Outer Zones.—As the workings of the North Pole, Eureka-

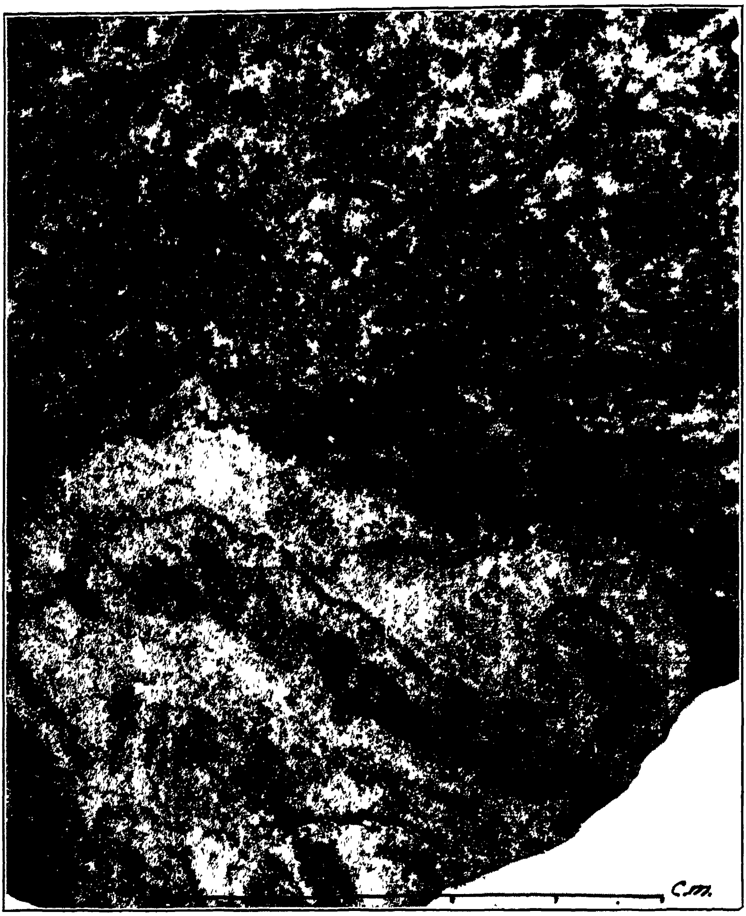
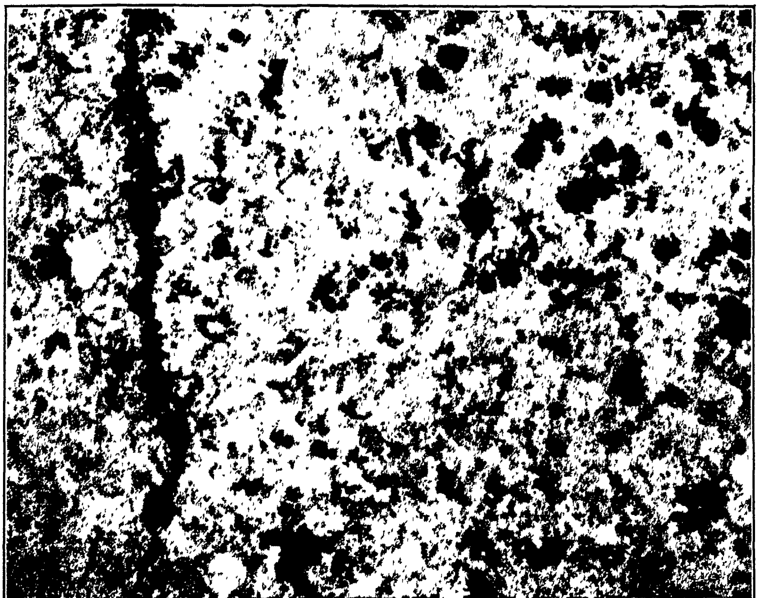


FIG. 10.—SPECIMEN FROM "MOTHER LODE," EUREKA-EXCELSIOR MINE.
Showing early layered quartz containing layers alternately rich and poor in sulfides (top and bottom of specimen) with later intimate mixture of quartz and pyrite. Position of thin section shown in Figs. 11 and 12 is indicated at upper left side.

Excelsior, and Columbia mines are connected, it is clear that they exploit the same vein. This vein has assuredly been explored for a distance of 12,000 ft. and to a depth of 2500 ft. below the highest point on the out-



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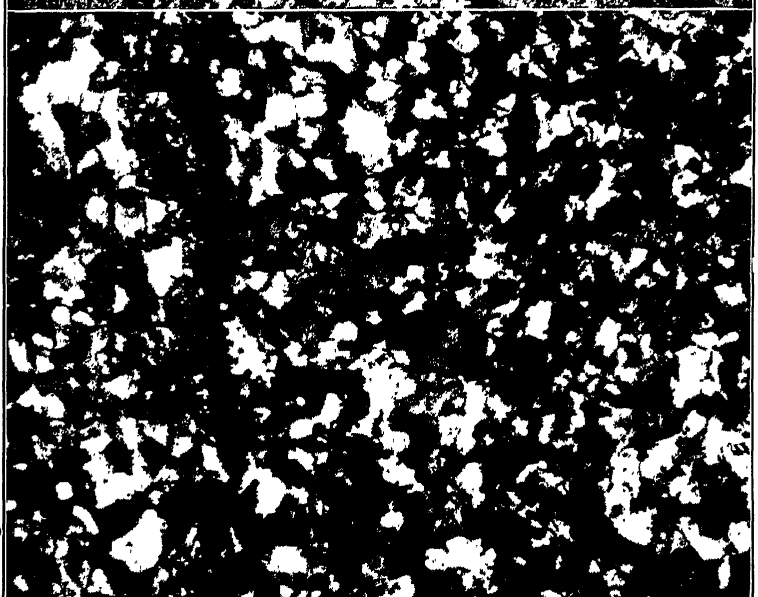

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FIG. 11.—SPECIMEN FROM "MOTHER LODE," EUREKA-EXCELSIOR MINE. × 30.

Showing intimate mixture of granular quartz and pyrite. Most of the pyrite forms minute detached grains, but a part forms distinct veins.

FIG. 12.—SAME SPECIMEN AS SHOWN IN FIG. 11, WITH CROSSED NICOLS. × 30.

Pyrite grains are associated with fine-grained quartz; coarse-grained quartz is free of sulfides.

crop. All the following statements concerning the features of the vein are based upon examinations made in 1908, 1914 and 1915, as the only workings now accessible are small parts of the North Pole and Eureka-Excelsior mines. The drifts from the Columbia and Eureka-Excelsior shafts are under water.

The composite character of this vein has been recognized by all who have described it.[17] In a broad way, it is made up of four or more strands of quartz and silicified argillite, separated by strands of gouge or sheared argillite. In several places in the North Pole mine the width of the vein between the outer walls attains 100 ft. and in one place 150 ft. Between these walls much of the material is argillite breccia in several stages of silicification, cemented by white quartz (Fig. 2). At one place, 1300 ft. from the mouth of No. 3 tunnel in the North Pole mine, there is a strand of 40 ft. of clear white vein quartz without residuals of argillite breccia. It is clear by inspection that sulfide minerals are sparse in most of the silicified breccia, and the records of mining and milling prove that such material is of very low grade—too low a grade, in fact, to be extracted with profit. The part of the vein that has been profitably mined in the past consists of one or two strands of quartz, distinctly separated from the rest of the vein by seams of gouge or by fractures. That part also contains more of the sulfide minerals, pyrite and arsenopyrite, than the other parts. So far as the evidence could be obtained, as late as 1915, this sulfide-bearing strand was the latest to be added to the vein.

Detailed examination of the part of the vein that has been mined shows that it also possesses a variety of complex structural features. In places, notably at the face of the Excelsior tunnel, 1550 ft. northeast of the shaft, an early white quartz has been thoroughly crushed, and the sulfide minerals, largely pyrite and arsenopyrite, have been deposited wholly in the finely crushed quartz. A much commoner kind of vein structure is found in the workings on the Excelsior and North Pole claims; in fact, so far as examinations since 1914 show, it appears to be characteristic of the part of the vein that lies northeast of the Eureka-Excelsior shaft, say 5000 ft. long. The peculiar features of this part of the vein are crusted layers of quartz, ¼ to 1 in. thick, alternating here and there with layers of dolomite and minute grains of sulfide minerals. Locally these layered growths were broken before succeeding layers were deposited (Fig. 10). The final material is a spongelike mass of quartz and sulfides in which the small aggregates of crystalline sulfides are dispersed unconnected (Figs. 11 to 14). Almost uniformly the nuclear sulfides are embedded in fine equigranular quartz and outward from these

[17] W. Lindgren: *Op. cit.*, 660.
J. T. Pardee: *Op. cit.*, 91–92.
J. T. Pardee and D. F. Hewett: *Op. cit.*, 84–92.

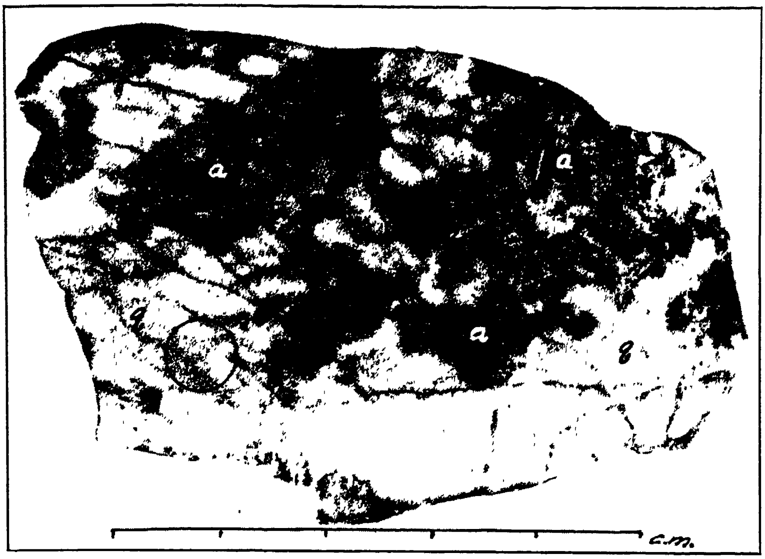

FIG. 13.—SPECIMEN FROM "MOTHER LODE," NORTH POLE MINE, INTERMEDIATE VEIN ZONE:

Showing irregular spongelike aggregates of minute grains of arsenopyrite *a* enveloped in quartz *q*. Specimen collected from strand 7 shown in Fig. 2. Position of thin section shown in Fig. 14 is indicated by circle.

FIG. 14.—THIN SECTION FROM "MOTHER LODE," NORTH POLE MINE. × 10.

Shows sporadic distribution of aggregates of fine-grained quartz crystals, outward from which coarse crystals grow. Sulfides, largely arsenopyrite, are confined to fine-grained areas.

nuclei much larger crystals of quartz extend radially. These seem to have grown until they interfered with similar coarse crystals growing outward from other sulfide nuclei near by. The crusts of quartz, dolomite and sulfides may be most readily explained as the materials successively laid down on free surfaces by solutions of changing character or concentration. It does not seem possible, however, to apply this explanation to the sponge-like mass of quartz and sulfides. If we dismiss the idea that the sulfides were laid down in a pre-existing rock which was later replaced by quartz, for this seems quite impossible, the most plausible explanation seems to be one that assumes the sudden cooling of a concentrated and slightly coherent solution or gel of sulfides and quartz, much as is contemplated by the hypothesis of ore magmas as proposed by Spurr.[18] It should be noted here that material of this class uniformly contains much more gold than the rest of the vein—commonly $50 to $150 to the ton. The state of the gold in this material has not yet been determined; that it is free gold seems very doubtful.

Columbia Mine, Outer Zone.—Most of the structural features noted in the North Pole and Eureka-Excelsior mines have been found in the Columbia mine, but a distinct difference lies in the much lower sulfide content of the vein in the Columbia. The recovery of free gold by amalgamation from ores has been uniformly higher in the Columbia mine than in the adjacent Eureka-Excelsior and North Pole mines.

Buckeye Mine, Outer Zone.—The Buckeye mine explores a group of nearly parallel veins in a carbonaceous variety of argillite. In most places the veins display several ramifying strands, each of which is made up of angular argillite fragments replaced and cemented by quartz. In thin section the argillite fragments show clearly as angular patches containing disseminated carbon in the midst of quartz, which is itself crushed and recemented by quartz. The percentage of sulfides is low.

Summary of Mineralogy and Texture of Ore Deposits around Bald Mountain Batholith

The only features that seem worthy of emphasis at this place concern the order and manner of deposition of the minerals. Among the sulfides, where pyrite and arsenopyrite are associated, pyrite is uniformly the earlier; in places the pyrite was brecciated before the deposition of arsenopyrite (La Bellevue), but generally the process of deposition was continuous (Independence, North Pole, Eureka-Excelsior). On the other hand, after pyrite and arsenopyrite were deposited, a period of brecciation generally intervened before the other sulfides were deposited. The order of the subsequent sulfides seems to be sphalerite, chalcopyrite,

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[18] J. E. Spurr: The Ore Magmas. New York, 1923. McGraw-Hill Book Co.

tetrahedrite and galena, although the exact relations of the first two are obscure.

The record of individual lodes indicates that the deposition of quartz has been a repeatedly interrupted rather than a continuous process, although all the lodes do not present similar records. Viewed broadly, the earlier quartz cements and to a large degree replaces angular fragments of the argillite that forms the walls. In some places the later quartz is deposited in new distinct strands; in others it cements a breccia of the earlier quartz. In most of the second group of lodes the later quartz has been added in the form of veins or cavity fillings in the pre-existing quartz, but in one place it has been deposited by enlargement of the earlier fragments so that the new and the old quartz are optically continuous.

In nearly all the places examined it seems clear that most of the quartz that forms the veins was deposited before any of the sulfides. Some veins record the deposition of quartz in the midst of and after the sulfide minerals, but the quantity is not large. It seems fair to assume, although it cannot be proved, that throughout the region there was an early stage following the intrusion of the quartz diorite, when all or most of the veins contained quartz with little if any sulfides, and that the sulfide minerals appeared in all the veins at a distinctly later epoch.

Thus far native gold has been seen in specimens from only a few of the veins surrounding the Bald Mountain batholith (Buffalo, Belle of Baker, Columbia, and North Pole), although it is reported from many other veins. In these veins the gold is either associated with the later sulfides or occurs in cracks in the quartz. It seems highly probable that the gold was deposited in the veins near the end of the epoch of mineralization. In this respect the veins of this region conform with the trend of interpretation in many regions studied in recent years (see pp. 343 and 344). So far as the writer can determine, the veins of the Sumpter region record much more fracturing of the pre-existing quartz than any others in the United States that have been studied.

Zonal Relations around Bald Mountain Batholith

The features of the lodes that reflect zonal relations around the intrusive quartz diorite include (1) the kind, crystalline state and quantity of the sulfide minerals present; (2) the percentage of gold recoverable by amalgamation, which seems to indicate the extent to which the gold is free or combined and the size of the particles. The data concerning the kind and crystalline state of the sulfide minerals are based entirely upon observation during the field work. The data concerning the quantity of sulfide minerals and the percentage of gold recoverable are based largely upon the records of milling operations in the district, which are summarized in Table 2. It is believed that these data show that the veins nearest

the intrusive center contain abundant pyrite and arsenopyrite and somewhat less sphalerite, chalcopyrite and galena in the form of coarsely crystalline grains, and that successive outward zones contain the sulfide minerals in decreasing quantity and in minute crystals; further, that the unweathered vein material near the intrusive center contains very little free gold, or, if the gold is free, it is extremely fine grained, and that in the successive outward zones the free gold increases steadily until in the outermost zone the gold contained is largely free and easily susceptible to amalgamation. Of course, oxidation has liberated the gold so that surface material from each of the zones readily yields most of its gold, but as the effect of oxidation extends only to shallow depths, it is not difficult to interpret its effect. Tellurides of gold, as well as the telluride of silver, hessite, have been found in several places, notably the North Pole mine, but it would be difficult to prove that this is the state of combination of the gold in mines of the inner zone.

Viewed areally, three distinct but not sharply separable zones may be recognized around the Bald Mountain batholith of quartz diorite. Of these, the inner zone seems to be best defined; it contains La Bellevue, California, Imperial Eagle, Highland, Maxwell, Baisley-Elkhorn and other mines (Fig. 1). The veins of La Bellevue mine lie in metamorphosed sedimentary rocks; those of the California and Imperial Eagle lie in the intrusive rock; the Highland in the sedimentary rocks and the Maxwell and Baisley-Elkhorn cross the contact. In other words, the mineral make-up of the zone is not dependent upon the local enclosing rock. Each of these veins contains the same assemblage of sulfide minerals, but there are local variations in the proportions of the sulfides. For instance, galena is uncommon in the Imperial Eagle veins but is abundant in the Highland. Although gold is the most valuable mineral in each of the veins, no vein yields more than 20 per cent of its gold content from unweathered material by amalgamation. As determined by milling processes, which are largely dependent upon gravity, the percentage of sulfide minerals in these veins is high, so that the ratio of concentration (tons of crude ore required to make a ton of concentrate) ranges from 5:1 to 10:1. The features described explain some of the problems that these mines have encountered. As the ores do not yield their gold by amalgamation, it has been necessary to resort to some process of concentration, mechanical or roasting and solution, but as the ratio of concentration is low, the concentrate from average ores commonly is worth only $25 to $40 a ton, and as the mines present difficult problems of transportation, it has rarely been possible to operate at a profit, especially in recent years. Although orebodies are known to exist in each of the mines mentioned above, the hope of successfully exploiting them seems to lie in perfecting processes that will recover the gold or yield a concentrate that is rich enough to stand transportation charges to the smelters.

It should be noted here that a few veins have been explored within the main mass of the intrusive quartz diorite, inside the inner zone indicated on Fig. 1. These are simple quartz veins which uniformly contain a little pyrite and chalcopyrite but no other sulfides. None have been explored extensively, and there is no record of production from them.

The Monumental veins are almost unique in that although in the kind and percentage of the outstanding minerals they resemble those of Cable Cove, which include the California and Imperial Eagle, the most valuable metal is silver rather than gold. The Ben Harrison veins in the Greenhorn intrusive yield material much like the Monumental, and in them silver greatly exceeds gold, but, unlike the Monumental, they contain much stibnite.

The intermediate zone is less well defined and contains fewer veins than in the inner zone. The Independence vein on the west and the northern part of the "Mother Lode" worked in the Eureka-Excelsior and North Pole mines are considered typical. In these veins, pyrite and arsenopyrite are the most abundant sulfides; they are uniformly finely crystalline, and the proportion of sulfides is indicated by the ratio of concentration, which ranges from 20:1 to 40:1. Of the gold which they contain, from 5 to 20 per cent. has been recovered by amalgamation. Common grades of ore, when concentrated by established gravity methods, therefore, yield a high-grade concentrate that can stand high transportation charges to smelters.

The Buffalo veins lie in the middle of the intermediate zone and yield material intermediate in character between that of the inner and outer zones. They contain all the common sulfides, in part coarse and in part fine, and under treatment by fine grinding and flotation the ratio of concentration is about 10:1.

The outer zone includes the Cougar, Ibex-Bald Mountain, Belle of Baker-Mammoth, Climax and Mayflower veins and that part of the "Mother Lode" explored in the Columbia and Golconda mines. These veins contain only a few varieties of sulfide minerals, which are rather uniformly fine grained, and the quantity is small. Good records of recovery under concentration are not available, but inspection shows that the ratio is as low as that of the veins in the intermediate zone, if not lower. Further, these veins yield a larger part of their gold, commonly 40 per cent. or more, by amalgamation.

Although most of the veins that lie in each of the three zones indicated on Fig. 1 are similar, a few veins have been explored of which the mineral make-up differs from those that seem to characterize the zone. For instance, the Blue Ribbon vein, northeast of the Buffalo mine, is a simple quartz vein with a little pyrite, which readily yields its gold by amalgamation and therefore resembles those of the outer zone rather than veins near by that are high in sulfide minerals. So far as the writer can ascer-

tain, however, the veins that are unlike those characteristic of each zone are very few. Veins having the make-up of those of the outer and intermediate zones are found here and there in the inner zone, but none characteristic of the inner zone occur in the intermediate and outer zones.

Ore Deposits Zonally Disposed around Greenhorn Mountain Batholith

Ben Harrison Mine, Inner Zone.—The width of the vein explored in the Ben Harrison mine largely ranges from 2 to 8 ft., of which one-quarter to one-half consists of several strands of quartz in the midst of sericitized quartz diorite. The vein, therefore, is much like the Monumental vein, near Granite. Thin sections of the quartz vein material show that an early barren quartz was crushed and arsenopyrite, pyrite, sphalerite, tetrahedrite and stibnite were deposited in the breccia. Later a barren white quartz cemented the crushed material and sulfides.

Snow Creek Veins, Intermediate Zone.—The principal Snow Creek vein is made up of a series of overlapping lenses of glassy white quartz, 4 to 8 ft. wide. Among the quartz veins thus far found in altered gabbro in this district, it is the widest. The quartz contains phantoms of angular fragments of gabbro and sporadic grains of chalcopyrite and galena. In part the quartz has replaced the gabbro. Near by in serpentine in the footwall, pyrite, chalcopyrite and galena, deposited in the order given, are disseminated through brecciated dolomite veins.

The Snow Creek vein seems to be typical of a number of veins in the Greenhorn district, of which the Banner, Banzette, Phoenix, Gold Bullion, Listen Lake, Dodo, Rabbit, and Humboldt have been examined. Most of these lie either in altered gabbro masses or along the contact of such masses with the later serpentine, but a few lie in serpentine. Although sulfides of lead and copper are generally present in the veins, the gold content is the principal source of value, and the average grade of the ore is higher than that of the veins in argillite farther east.

The Golden Eagle vein seems to be typical of another group of veins near Greenhorn, sporadically distributed near the quartz veins described above. This group includes the Muscatine, West Side, Diadem, Don Juan, Eccentric and others. These lie wholly in serpentine or along contacts of serpentine and gabbro. They are rather simple, narrow veins, which generally include a single strand of dolomite with sporadic sulfides that in part replace the carbonate and in part fill fractures.

The veins referred to briefly above all lie in the igneous rocks—gabbro, serpentine and diorite—intrusive into the sedimentary argillites. In an outer zone there is another group of veins in the argillite. This zone seems to be traceable from the Red Boy mine, on the north, to the Bonanza mine, 7 miles southeast. It includes also the Blue Bird, Black Jack, Maid of Oregon, Royal White, Belcher, Golden Gate, Pyx, I. X. L.,

Bonanza and other mines. Most of these mines have not been worked during the past 15 years.

Zonal Relations around Greenhorn Mountain Batholith

In attempting to view the ore deposits of the Greenhorn and near-by areas in the light of zonal relations, the writer is handicapped by the lack of knowledge of the areal geology west of Sumpter quadrangle and of many small mines in that region. One is at first struck by the lack of zones closely similar to those recognized around the Bald Mountain batholith. There seems to be no equivalent of the well-defined zone displaying the group of veins that contain high percentages of sulfides. Most of the veins near Greenhorn, whether made up largely of quartz or of dolomite, contain pyrite, chalcopyrite and galena, but the ratio of gangue to sulfides is high and arsenopyrite is conspicuously lacking. So far as records are available, the gold which they contain is readily susceptible to amalgamation, but few have been explored below the zone of oxidation. Excepting those with dolomite gangue, these veins broadly resemble the group found in the middle zone near Bald Mountain.

The veins in the outer zone, however, have considerable resemblance to those of the outer zone surrounding the Bald Mountain batholith. They contain a small percentage of sulfides, of which pyrite is the most abundant, and 70 per cent. or more of the gold present has been recovered by amalgamation.

Relation of Textural Features to Zonal Arrangement of Veins

The outstanding results of the study of the structural and textural features of the veins of the Sumpter region may be briefly summarized. Not only are most of the veins made up of several strands of quartz and sulfides, but almost every one that has been studied closely records numerous epochs of crushing each followed by cementation of the resulting breccia. Further, there is not only a rather definite sequence of the sulfide minerals, but all appear to have been deposited rather late in the history of the veins. Quartz has been deposited repeatedly both as distinct strands and as a filling in breccias of pre-existing quartz. The early quartz was capable of replacing the argillite country rock to a high degree and contained small quantities of sulfides, if any; by contrast, the later quartz did not so readily replace the argillite and was associated with considerable quantities of sulfides.

Probably when any large body of sedimentary rocks is intruded by an igneous mass each part of the sedimentary body tends to undergo a simple cycle of temperature—that is, it is cold before intrusion, rises to a maximum temperature at some time after intrusion, and then cools slowly. Variations from a simple cycle of temperature are common and seem to be related to complexities of the intrusion epoch. As ore deposition

largely takes place long after the intrusive rocks are solid, most of it undoubtedly occurs in the declining part of the temperature cycle. Without further speculation here concerning the process of ore deposition, the opinion may be expressed that in proportion as the intrusive history is simple and brief the veins that are formed near by should be simple filled fractures and their mineral content should display a simple zonal make-up.

This region presents the apparent anomaly of a simple intrusive history (at least, if the earlier intrusions of gabbro and serpentine are ignored), complex vein structure and texture, and a tendency toward simple zonal arrangement of the mineral make-up. The only explanation for this anomaly that can be offered here is based upon the interpretation that although the principal intrusive mass is mineralogically simple and homogeneous, it is also very large. The many movements recorded by the veins may reflect the complex local adjustments of the parts of so large a mass during cooling.

It may be well to raise for brief consideration here the question as to the bearing of the character of the veins upon the manner in which they were formed. Until recent times it has been assumed that quartz veins represented the material deposited from rather dilute aqueous solutions rising from considerable depth along fractures, the walls of which were either forced apart by the pressure exerted by growing crystals or by the pressure that forced the solutions to rise against the great weight of the rocks. Recently Spurr[19] has proposed the hypothesis that most quartz, as well as some other mineral veins, represents congealed dense liquids.

The evidence obtained during this investigation indicates that, to an unusual degree, the veins have been built up by successive deposition of quartz in breccias—first breccias of country rock and thereafter breccias of the pre-existing quartz. On the other hand, some veins yield evidence (Figs. 10 to 14) that a part of the quartz represents a hardened dense liquid, much as Spurr has postulated. In the final report on this region the matter will be presented and discussed in greater detail.

Summary of Zonal Relations

It is believed that most of the veins of the Sumpter quadrangle, especially those which have been most extensively explored, fall rather readily into a simple zonal scheme. The zones are shown by the kind, quantity and state of aggregation of the minerals present—the sulfides as well as the gold. The zones are best displayed around the Bald Mountain batholith, and it is thought that the less definite arrangement around the Greenhorn intrusive is due in part to the different character of the rocks near by that contain the veins.

[19] J. E. Spurr: The Ore Magmas. New York, 1923. McGraw-Hill Book Co. Discussion on Magmas, Dikes and Veins. *Trans.* A. I. M. E. (1926) **74**, 99–115.

If this generalization is accepted, it possesses more than academic interest as a feature of the veins. If it is true that the roughly horizontal surface cut by erosion across the contact of the Bald Mountain intrusive reveals the fact that these portions of the veins worked near that surface have certain features that fall into zones, it is highly probable that further exploration of the veins in depth will show changes in mineral make-up in accordance with those zones. This inference contemplates that the zones may be likened to the concentric layers of an onion whose center corresponds to the deeply buried center of the intrusive body. Erosion has cut a roughly horizontal surface above the center, which displays the several zones as rudely circular bands. Explorations vertically downward at any place will pass through the same zones that are revealed on the surface progressively inward toward the center of the main batholith. From this it follows that veins like that explored by the Cougar and Columbia mines will tend to pass vertically downward into veins resembling the Independence, Buffalo, Eureka-Excelsior and North Pole. In other words, veins that contain a low percentage of sulfides and yield their gold readily by amalgamation will tend to show with increasing depth an increasing percentage of sulfides and decreasing yield of free gold. A pertinent question concerns the depth at which such changes should be shown in mine explorations. The horizontal zones that have been recognized range from 1 to 2 miles in width, so that if the zones projected in space inclined 45° outward, depths of 1 to 2 miles would be required to show a similar change. Probably the zonal layers dip less steeply than 45°, and changes should be observed within a few thousand feet or less. According to R. S. Amidon, recent explorations at the Buffalo mine, at a depth of 400 ft. below the outcrop, show a steadily increasing percentage of sulfides, and according to John Arthur, who recently operated the Eureka-Excelsior mine, a similar increase was noted.

Comparison with Similar Districts

In recent years geologists investigating the detailed relations of gold and sulfide minerals in quartz veins have almost uniformly recognized evidence tending to show that the veins were first filled with quartz that was barren of gold and sulfides and that these minerals were deposited late in the history of the vein, with or without associated quartz, after one or more intervening epochs of crushing. As early as 1895 Lindgren[20] recognized this feature as characteristic of California quartz veins in general, and recently Knopf[21] has recorded it in the Mother Lode mines,

[20] W. Lindgren: Characteristic Features of Gold Quartz Veins. *Bull.* Geol. Soc. Amer. (1895) **6**, 228–229.

[21] A. Knopf: The Mother Lode System of California. U. S. Geol. Survey *Prof. Paper* 157 (1929) 25–26.

California. Similar relations have been recorded at Alleghany, Calif.,[22] Kirkland Lake and Porcupine, Ontario;[23] Grass Valley, Calif.;[24] in numerous Canadian districts,[25] and elsewhere.[26]

Probably the most valuable practical conclusion that can be deduced from these observations concerns the nature and distribution of the ore shoots. Veins of clear white quartz free of sulfide minerals rarely can be mined as gold ore; gold and sulfides are largely confined to the parts of the veins that have been crushed or the strands of quartz that have been deposited late in the history of the vein. It should be the aim of mining operations to recognize and follow such parts of the veins.

The writer cannot find that the kind of zonal distribution of the vein contents shown in the Sumpter district has been recorded elsewhere. When first announced by Spurr in 1907, the hypothesis of zonal distribution postulated not only that the metalliferous deposits of a region (largely the veins) are closely related to the igneous intrusions but that groups of metals tend to outcrop in zones around the intrusive center, and that veins explored in depth tend to show similar successive vertical zones of the metals. It is probably correct to state that since the hypothesis was proposed, although a tendency toward zonal arrangement of metalliferous deposits has been noted widely, only rarely have more than two or three of the six zones suggested by Spurr been recorded, and in many places no such tendency is apparent.

Outlook for the District

Many elements deserve consideration when an attempt is made to estimate the outlook of a metal-mining district, and, in a broad way, the trustworthiness of an estimate is determined by the degree to which the necessary information concerning these elements is available. Among the geologic elements that should be considered are the relation of the ore-bearing fractures to the outstanding geologic features of the region, and any similarities to other productive regions that may be noted; the

[22] H. G. Ferguson: Lode Deposits of the Alleghany District, California. U. S. Geol. Survey *Bull.* 580 (1915) 153–182.

H. G. Ferguson and R. W. Gannett: Gold Quartz Veins of the Alleghany District, California. A. I. M. E. *Tech. Pub.* 211 (1929) 24–29.

[23] A. G. Burrows and P. E. Hopkins: The Kirkland Lake Gold Area. Ontario Dept. Mines *Ann. Rept.* (1923) **32**, Pt. 4.

A. G. Burrows: The Porcupine Gold Area. Ontario Dept. Mines *Ann. Rept.* (1924) **33**, Pt. 2.

[24] E. Howe: The Gold Ores of Grass Valley, California. *Econ. Geol.* (1924) **19**, 604–605.

[25] G. W. Bain: Structure of Gold-bearing Quartz in Ontario and Quebec. A. I. M. E. *Tech. Pub.* 327 (1930) 44.

[26] C. D. Hulin: Metallization from Basic Magmas: Univ. Calif. Dept. Geol. Sci. *Bull.* (1929) **18**, 233–284.

proved length, width and depth of the veins; and the relations of the minerals to horizontal or vertical zones, due weight being given to the possible presence of a zone of surface enrichment and its relation to the relief of the region. Obviously, the distribution and yield of placers should be considered in any region of gold-bearing veins. Several technical factors must also be taken into account—for example, the quantity and grade of the material recovered underground by the methods and the efficiency of extraction of the processes in use at the time of exploitation. For a comprehensive judgment, consideration should be given also to some economic elements, such as the cost of the many materials entering into mining and the value of the products.

From both the extent of underground explorations in length and depth and the relations of the fractures to the geologic features of the region, it is clear that the veins of the Sumpter region have more than average persistence horizontally, and it seems that they should persist vertically much deeper than they have been explored thus far, especially those veins that are grouped around the Bald Mountain batholith. Considerable information, largely unpublished, is at hand concerning the grade of ore mined, the yield and grade of concentrate shipped, the extent of shoots and assay maps. From these data it seems clear that the ore from the mines that have supplied a very large part of the production of the district has contained metals, largely gold, worth from $5 to $15 to the ton. Also, it seems clear that some of the ore recovered from the weathered parts of many veins, say 50 to 150 ft. below the surface, was richer than the average recovered below, owing more to the removal of metallic sulfides by solution than to solution and redeposition of the gold. This enrichment has been more noticeable in the veins of the inner zone surrounding the Bald Mountain batholith than in the others, because the former contain more sulfide minerals. If only the deeper unweathered parts of the veins are considered, better grades of ore have been obtained from the veins of the intermediate and outer zones than from those of the inner zone. This difference appears to be due largely to the higher content of heavy sulfide minerals in the inner zone and consequent greater weight of the ore per unit of volume.

So far as size of the oreshoots is concerned, whether indicated by size and distribution of stopes or by assay maps, there can be no doubt that the "Mother Lode" explored by the Golconda, Columbia, Eureka-Excelsior and North Pole stands in a class by itself. Compared with this vein, the shoots of all the other known veins of the district are much smaller in horizontal plan, although it is entirely possible that they may persist as deep. Not only is the total production of the "Mother Lode" group of mines several times greater than that of all the mines of the inner zone, but the probable profit from this group over the entire period of operation has shown a still greater difference. In fact, it is doubtful

whether any mine in the inner zone, unless it is the Baisley-Elkhorn, has shown a profit over any 10-year period of operation.

The period of greatest activity in this region extended from 1895 to 1910, although a few mines have been productive intermittently since 1910. The records of operation from 1895 to 1910 indicate uncommonly high losses in treatment, but since 1915 there have been great improvements in the processes applicable to such ores, notably flotation. Only three mines, however, the Eureka-Excelsior, Buffalo and Ben Harrison, have been able to apply these improved processes. From what is known of the grade and extent of the ores encountered in the Columbia, Eureka-Excelsior, and North Pole mines when they were last worked it seems clear that a consolidation of this group offers the best chance for a profitable operation in the district. On the other hand, although shoots of ore of average grade are known to exist in mines in the inner zone, their sulfide content is so high that unless refinements of flotation or similar processes can be perfected which can selectively recover most of the gold from the sulfide minerals and quartz, their operation will continue to be hazardous.

Of the veins associated with Greenhorn intrusive that exploited in the Bonanza mine has been most productive, with the Red Boy and Ben Harrison following. Even though the Bonanza veins have been explored 1250 ft. below the outcrop, and the grade of ore has been higher than the average, the shoots have been small in horizontal plan, and this statement appears to be true of most if not all of the veins associated with the Greenhorn intrusive.

The Pao Deposits of Iron Ore in the State of Bolivar, Venezuela*

By Ernest F. Burchard,† Washington, D. C.

(New York Meeting, February, 1930)

The Pao deposits of iron ore are in the State of Bolivar, Venezuela, near Upata, about 30 miles south of the Orinoco River and 90 miles east of Ciudad Bolivar. They were discovered 4 or 5 years ago and have not yet been developed to the stage of mining.

In March–April, 1929, the writer devoted a week to the examination of these deposits. The maps, drill records and chemical analyses presented herewith were derived from original sources, and the field photographs were made by the writer. The petrographic and metallographic data are from notes by C. S. Ross and M. N. Short, of the U. S. Geological Survey, who have kindly collaborated in these special phases of the study.

The Iron Orebodies

Topography and Accessibility

The deposits are in a region of low mountains representing northern outliers of the Guayanan Highlands. These mountains reach altitudes of 1800 to more than 2000 ft. above sea level, as determined by aneroid barometer. The iron ore is found on mountain crests that reach altitudes of 1915 to 1930 ft., and may be followed down precipitous slopes for 200 to 300 ft. or more. The greatest vertical range in outcrop is nearly 400 ft., and drilling has indicated that the ore extends to this depth below the surface in one place, at least. The superior hardness of the iron ore accounts for the preservation of the local mountain crests with the intermediate valleys eroded in softer granitic rock.

The deposits of iron ore are reached by a private road for automobiles branching northeast from the Ciudad Bolivar-Upata road, a distance of about 12 miles. This road was used in 1928–29 by trucks for transportation of food and camp supplies, gasoline, drilling equipment, etc., but was not suitable for the transportation of iron ore. Between Ciudad Bolivar and the iron-ore area the western two-thirds of the automobile road passes through nearly level, open country known as the savanna, largely used for grazing cattle. About 96 km. (60 miles) east of Ciudad Bolivar at

* Published by permission of the Director, U. S. Geological Survey, and through the courtesy of Mr. F. D. Pagliuchi, who has brought the deposits to their present stage of development.

† Mining Geologist, U. S. Geological Survey.

Caruachi the road reaches Rio Caroni, a stream that is both wide and deep even in the dry season, across which automobiles are carried on a small ferry boat propelled by a gasoline launch. At low stages of the water there are stretches of sand on each side of the river that are difficult for cars and trucks to pass without assistance.

The region is rough and mountainous in the vicinity of the iron-ore deposits and thickly overgrown with forest trees, vines and brush. A tree known locally as "copey," having rather soft, brittle wood, glossy dark green leaves, and a milky sap, thrives on the iron ore, and the occurrence of this plant is of assistance in exploration for the orebodies.

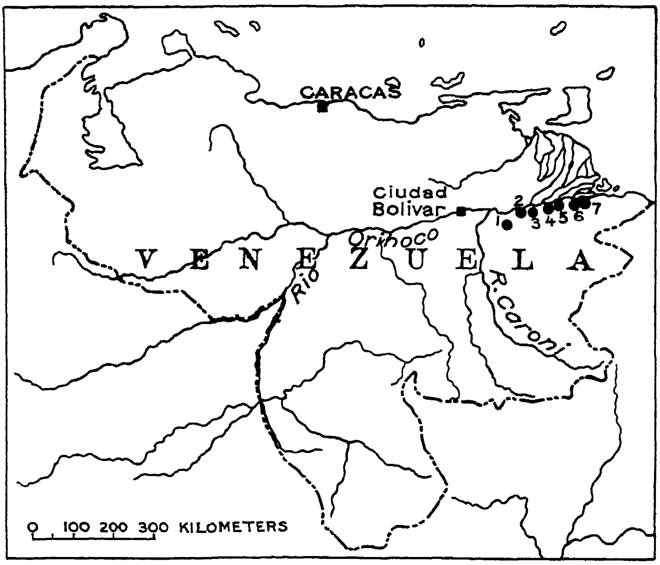

FIG. 1.—MAP OF VENEZUELA SHOWING DISTRIBUTION OF IRON-ORE DEPOSITS SOUTH OF ORINOCO RIVER.
1, Pao; 2, Los Castillos; 3, Piacoa; 4, Santa Catalina; 5, Sacuroco; 6, Manoa (Imatáca); 7, La Escondida.

To take the ore to the Orinoco River will necessitate building a standard gage railroad northward to San Felix, a distance of about 30 miles. It is reported that the route for this line has been explored and that it is a feasible route, mostly down grade to the river. Some large waterfalls, situated on Rio Caroni about 6 miles from San Felix, are reported to be capable of furnishing in the midst of the dry season not less than 100,000 hp. Power from this source would be very helpful in operating the railroad and mining machinery, in the pumping of water, and in other services to a mining community.

To utilize the ore in North America or Europe will necessitate moving it down the Orinoco River, probably in shallow-draft boats in order to cross bars affording not more than 6 to 8 ft. of water and transshipment

by ocean-going vessels from some Venezuelan or Trinidad port on the Gulf of Paria.

Distribution

Twenty-two mining claims, covering an area of about 6 by 7 miles, comprise the deposits of iron ore in the Pao region. The top of the mountain on which the largest deposits of iron ore occur presents a roughly

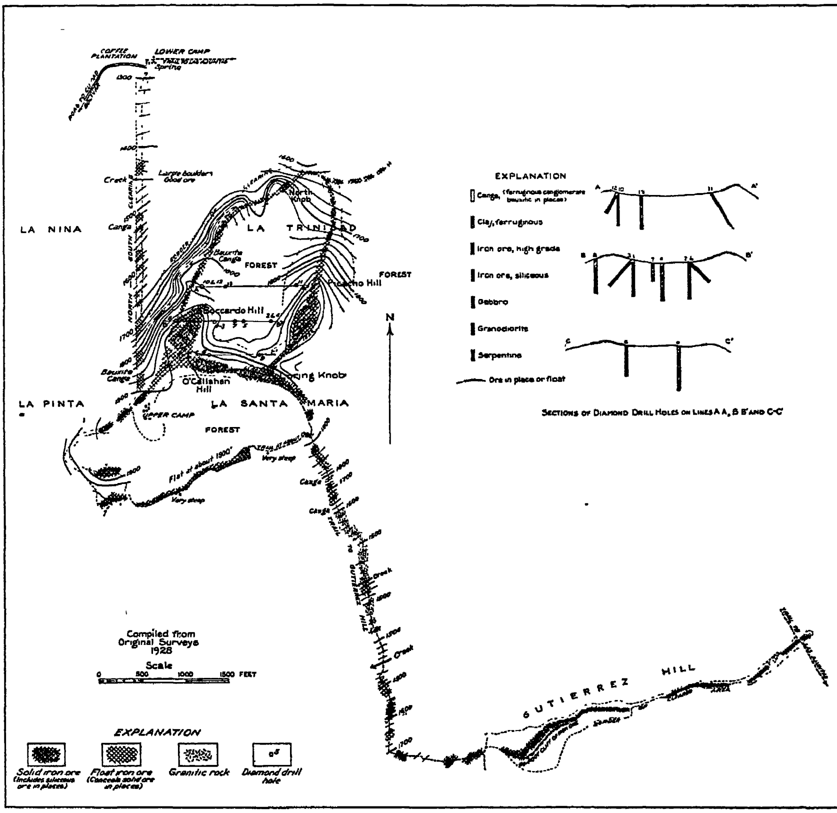

FIG. 2.—MAP OF PAO IRON-ORE AREA, VENEZUELA, WITH CROSS-SECTIONS THROUGH DIAMOND-DRILL HOLES.

elliptical outline, extending about 1500 ft. or more east to west and about 4500 ft. south to north. The southern portion of the area is nearly all at a level of about 1900 ft. above the sea; the central portion shows several residual knobs and ridges that rise a little above the general level and form a horseshoe-shaped border on the south of a small basin which slopes toward the north. (See Figs. 2 to 9.) Taken altogether, the trace of the outcrops of the iron ore in the principal area forms roughly a figure 8, around which one may walk over iron ore almost continuously for about

2.3 miles. In addition to the ore comprised in this area there is a ridge known as Gutierrez Hill, lying about 0.8 mile southeast of the nearest ferruginous portion of the figure 8, along which masses of iron ore having widths of 50 to 300 ft. and heights ranging from 60 to 200 ft. may be traced at intervals for a distance of about 0.8 mile. About ¾ mile north of the main (Boccardo) area across a small valley through which the trail passes from the Pao Lower Camp to Las Ajuntas plantation, there are on the south slope and crest of the ridge other deposits of high-grade iron ore not shown on any available map.

This region has been explored very little and as it is only 4 or 5 years since the Pao deposits were discovered, it seems probable that other similar deposits of iron ore not yet known may exist in the heavily wooded ridges toward the east and southeast.

FIG. 3.—VIEW TOWARD ORINOCO RIVER FROM PAO IRON-ORE DEPOSITS, VENEZUELA.

Surficial Material

Float Iron Ore.—In addition to the large extent of ore in place on the outcrops of the orebodies there is a great deal of float ore in the form of large blocks, slabs, boulders and pebbles, which are residual from former higher extensions of the orebodies that have been reduced by weathering and erosion. Some of these masses of ore have reached their present position by gradually settling as the underlying ore was carried away. In some instances, because of the jointing of the masses on the outcrops, it is difficult to decide whether a block of ore is actually in place or not. The larger blocks probably have not moved far from their present position; pebbles and cobbles of ore show evidence of transportation by streams.

Canga.—In places the float ore, consisting of lumps of hematite and magnetite, has been cemented together, mainly by limonite, forming

an iron-ore conglomerate. This material is called *canga*, a name applied to similar conglomerate in Minas Geraes, Brazil. The surface canga is for the most part a mixture of fairly pure iron oxides, although in places it may contain some silica. Below the surface, however, there is found in most places a bauxitic canga which ranges from nearly pure, pisolitic bauxite to bauxite containing more or less hematite and limonite. In

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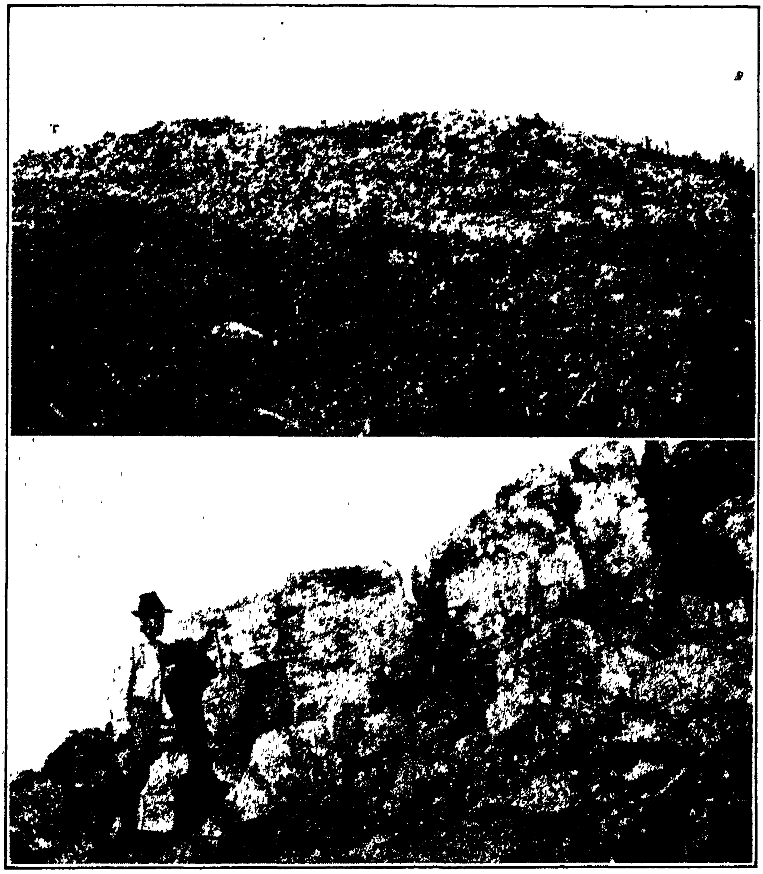

FIG. 4.—SOUTH END OF BOCCARDO HILL AT PAO, SURFACED WITH IRON ORE.
FIG. 5.—MASSES OF IRON ORE PRACTICALLY IN PLACE ON NORTH END OF BOCCARDO HILL.

places the bauxite crops out at the surface, as on the north-south line cleared for the telephone wire from the lower to the upper camp and on the east side of the narrow ridge northwest of drill hole 10. The thickness of the canga, most of which may be assumed to be bauxitic, ranges according to the drill records from 10 to 76 ft. In one hole, 115 ft. of good iron-ore canga was reported.

If the ferruginous bauxite can be washed and concentrated so as to remove the coarser iron contents, the bauxite may prove to be a valuable by-product, particularly if it is necessary to strip it from the deposits of iron ore in mining them by open-cut methods.

6

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Fig. 6.—Picacho Hill, surfaced with iron ore, viewed from Boccardo Hill.
Fig. 7.—Loose blocks of iron ore on north end of Picacho Hill.

Geologic Relations

In preparation for prospecting the deposits of iron ore, the jungle growth was cleared in 1927–28 from many of the most prominent outcrops and a few trails were cut from place to place. Although the vegetation has grown considerably since this work was done, much of the cleared surface was still visible in April, 1929, but otherwise the forest growth and soil cover were so thick as to preclude systematic examination of the rocks and in a brief visit little could be seen of the local geology except where clearings had been made. For this reason the knowledge of this subject is not as adequate as might be desired.

The country rock south of Rio Orinoco westward from the Guayanna Highlands to above Ciudad Bolivar is in general granitic. Liddle[1]

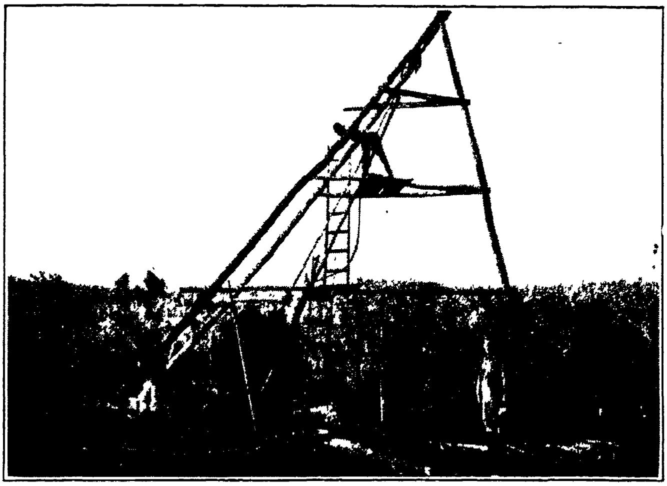

FIG. 8.—DRILLING IRON-ORE DEPOSIT AT PAO.

FIG. 9.—HUMP AND CLIFF OF IRON ORE ON GUTIERREZ HILL, PAO.

characterizes the rocks that constitute the floor of the Guayanan Highlands as granitic gneiss and gneissoid granite, and considers them pre-Paleozoic, probably of igneous origin, and as having undergone much metamorphism in places. He says that other metamorphic rocks, derived from Paleozoic sediments and consequently younger, are iron-bearing quartzites apparently faulted into the basement complex.

Typically rounded residual knobs of granitic rock are common along the road over the savanna east of Ciudad Bolivar and in the city a dome-

[1] R. A. Liddle: The Geology of Venezuela and Trinidad, 55. Fort Worth, Texas, 1928. J. P. McGowan Pub.

shaped mass of fairly fresh light-colored granite cut by aplite is well exposed in the hill west of the Cathedral. On the shore of Rio Orinoco in the western part of Ciudad Bolivar the granite is well exposed and included in it in places are masses of darker schistose rock. In many places along Rio Orinoco above Los Castillos and at the Rio Caroni ferry at Caruachi are masses of rock, weathered black, which appear to be dikes of basic igneous rock but upon close inspection are found to be of the typical granitic rock of the region.

Specimens of characteristic granitic rock from the Pao Lower Camp and from Las Ajuntas trail a few hundred yards east of the camp, examined in thin section by Dr. C. S. Ross, were found to be, respectively, quartz monzonite and granodiorite. There is some doubt whether these granitic rocks actually are of the pre-Paleozoic series indicated by Liddle because certain field relations indicate that they may have invaded and intruded the iron-bearing quartzose rocks, metamorphosing them to schist in places (Fig. 10). No direct contacts were observed between the iron-bearing deposits and the granodiorite, but the latter rock is well exposed at the Pao Lower Camp at levels 200 to 400 ft. lower than the ore and near the creek level on the trail from Pao, Upper Camp, to Gutierrez Hill, and the quartz monzonite is exposed along the Pao-Las Ajuntas trail northeast of Boccardo Hill.

Associated with and bordering the rich deposits of iron ore are beds that consist chiefly of banded quartz and iron oxide (Fig. 11). The proportion of iron oxide present varies greatly so that the rock ranges from a quartzose material with a few specks of iron oxide in widely separated bands to iron ore containing a few siliceous streaks, and a complete gradation between these extremes is shown. Most drill records show the quartzose ferruginous rock on the borders of the orebodies and in some places interbedded with the ore, so that it appears to be part of the series of iron orebodies. This quartzose iron-bearing rock resembles to some extent the Itabira iron-bearing siliceous rock that is associated with and grades into the iron ore in the large deposits in the State of Minas Geraes, Brazil. The iron-ore deposits in Minas Geraes are believed by those who have studied them thoroughly to be of sedimentary origin, and a comparison of certain features of the Pao deposits leads at first to the thought that they may be of similar origin. There are, however, features of dissimilarity that indicate a more complex history for the Pao ores.

Although no contacts between the iron ore and the granitic rocks were observed by the writer, drill records show that certain prospect holes passed from the ore into quartzose ferruginous rock and then into granitic rock. Where the ore is under a heavy cover, as in the basin between Boccardo and Picacho hills, it is shown by certain drill records to be overlain by soft serpentine which is in turn overlain by gabbro.

Dr. Ross examined pieces of cores from the drill holes, confirming the field identification of the gabbro (Figs. 12 and 13) and the serpentine, and expressed the opinion that the serpentine has been formed by alteration from dunite. The serpentine contains many dark specks composed of ilmenite and manganese oxide, but no chromite was found in it. The gabbro and dunite may be sill-like masses that were intruded into the quartzose ferruginous rocks, and that had an influence on the metamorphism and enrichment of the iron-bearing beds.

More or less bauxite is present near the surface. In a few places it is exposed but it is usually covered by soil and clay or float of fresh iron ore. Some of this bauxite is fairly pure, pisolitic material, but much of it contains specks and fragments of iron oxide and therefore becomes a bauxitic canga.

Form and Structure

The outlines of the bodies of iron ore have not been definitely demonstrated by drilling but from such records as are available and from the evidence afforded by outcrops they appear to be generally tabular in shape but swelling into a lenticular form in several places. Judging from the quantity of float ore present, the thicker portions of the orebodies formerly must have extended to much higher levels.

The attitude, or position of the deposits with relation to the surface, is that of steeply inclined beds or tabular masses. The material at the surface is much jointed, so that it is difficult to distinguish between joints and the planes that represent bedding or a corresponding natural separation of the material into layers. If the terms strike and dip may properly be applied to the material, it is found that the strike is extremely variable, for the bodies may be traced in almost every direction so as to form in one area, as suggested above, roughly the figure 8, but the strike of the greatest extent of orebodies is toward the north-northeast.

In the horseshoe-shaped area bordered by Boccardo and Picacho hills, the bodies of iron ore appear to dip toward the interior of the basin; that is, eastward on Boccardo, northward on O'Callahan and Loring, and westward on Picacho. If these dips flatten out the orebodies may come together and form a continuous sheet of iron ore at depths varying from a few feet to 300 ft. There is evidence that part of the basin is underlain by iron ore but more drilling will be necessary to make certain how far the ore area extends. (See sections of drill holes, Fig. 2.)

The bodies of iron ore extend in places to depths of more than 400 ft. below the surface, as determined by drilling, but as no hole has been drilled directly down the dip, or pitch, of the ore the maximum depth to which it may extend is not known. The difference in altitude between ore in place on top of Boccardo, Picacho and intervening hills and on

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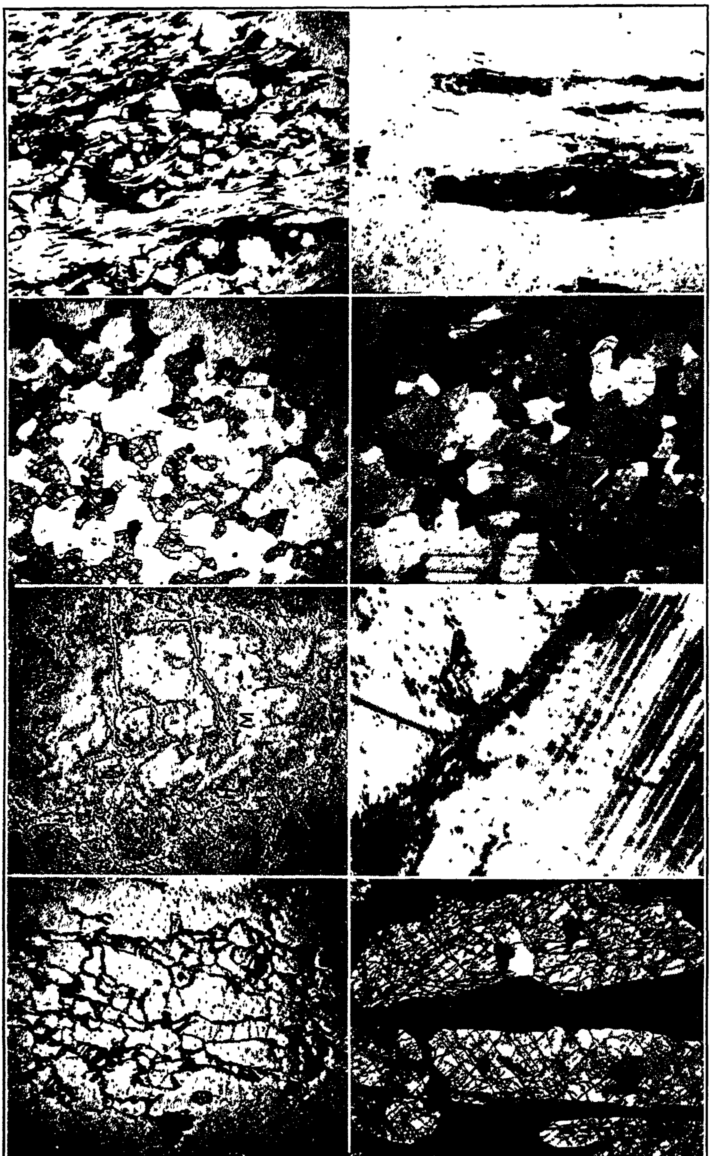

FIG. 10.—SPECIMEN NO. 47. THIN SECTION. IRON OXIDE (BLACK) IN SCHIST (WHITE). × 15.

FIG. 11.—SPECIMEN NO. 51. THIN SECTION. IRON OXIDE (BLACK) IN QUARTZITE (WHITE). × 15.

FIG. 12.—SPECIMEN NO. 34. THIN SECTION. GABBRO. × 15.
White areas are plagioclase feldspar; dark gray areas with well-defined cleavage cracks are pyroxene; black rounded areas are magnetite.

Figs. 10, 11 and 12 taken in ordinary light. Reduced 50 per cent.; original magnifications given.

the slope of the valley north of these hills is nearly 400 ft., and ore can be traced on the outcrop almost continuously between these places. There are places where precipitous cliffs and slopes show iron ore vertically for 100 to 200 ft., as on the west of Boccardo and southeast of the Upper Camp.

THE IRON ORE

Surface Appearance

Where fresh, the iron ore is predominantly a bright, steely gray, hard iron oxide of fine to coarsely crystalline texture. Large masses of ore such as form the cleared tops of Boccardo, Picacho and neighboring hills, on which no other rock is exposed, when viewed from a distance appear to be a massive light-gray gneiss. A faint banding may be detected in some of the ore, which suggests replacement or hydrothermal alteration of rock that originally possessed a banded structure. In places the ore crops out in massive blocks and cliffs but over much of the surface it is in large loose slabs, boulders, or blocks, as shown in Figs. 5 and 7. The ore has been much fractured and jointed and solution has proceeded underground along the joints so as to form cavities or channels. These openings have caused considerable trouble in drilling. In many places there are evidences of stream action in the form of well-marked pot holes in the surface of the ore, and well rounded pebbles and boulders are found composed of hard hematite covered with a film of limonite, all of which indicates stream action over the iron-ore area in times past.

Mineralogy

The mineralogy of the ore deposits is essentially very simple, as there is little besides iron minerals and quartz in the deposits, so that the

FIG. 13.—SPECIMEN NO. 34. THIN SECTION, FROM SAME SECTION AS FIG. 12. × 15. Taken in polarized light.

FIG. 14.—SPECIMEN NO. 1. POLISHED SECTION. HEMATITE (*H*) REPLACING MAGNETITE (*M*). × 100.

Replacement apparently proceeds along cracks and other open spaces. In polarized light all the hematite extinguishes simultaneously. The magnetite is probably all part of one crystal and the hematite has oriented itself along crystallographic directions in this crystal.

FIG. 15.—SPECIMEN NO. 17. POLISHED SECTION. COARSE CRYSTALLINE HEMATITE. × 85.

Photographed in reflected polarized light. Entire picture shows two grains of hematite divided by a veinlet of gangue running N.E.-S.W. across the picture. The upper left grain shows twinning in two directions of the unit rhombohedron. The lower right grain shows multiple twinning in one direction also of the unit rhombohedron.

FIG. 16.—SPECIMEN NO. 48. POLISHED SECTION. HEMATITE (WHITE) REPLACING QUARTZITE. × 35.

Parallelism of hematite is clearly seen, following direction of schistosity in specimen. Hematite for the most part occupies spaces between individual quartz grains and cracks in quartz.

FIG. 17.—SPECIMEN NO. 36. THIN SECTION. HEMATITE (BLACK) REPLACING CORUNDUM (GRAY). × 15.

Corundum shows well-defined parting planes. Ordinary light.

All reduced 50 per cent.; original magnifications given.

occurrence of a small mass of bladed corundum crystals found by the writer on the top of a cliff of iron ore about 0.25 mile southeast of the Pao Upper Camp seems interesting and significant. This corundum occurs within a folded mass of the iron ore, here mostly hematite, but in thin section the corundum appears to be earlier than the hematite (Fig. 17). The chemical analyses (Table 2) show that alumina is more abundant than silica in the ore from the diamond-drill core. Bauxite occurs on the surface of the ore in places, clearly a lateritic product. This may have some bearing on the segregation of the corundum. There is no limestone in the area and no contact metamorphic minerals are found in the ore. No sulfides were noted and, as shown by chemical analysis, the ore is low in sulfur.

The ore is predominantly hard, crystalline hematite, but most of it carries magnetite, irregularly distributed within the mass or distributed in fairly definite bands. In some fragments of ore, magnetite may be detected by the deflection of the compass needle, but others tested in the same manner show little or no magnetic attraction. Polished surfaces of some pieces of the ore show grains of magnetite embedded in the hematite while others show none. A piece of ore from a drill core at depth of several hundred feet showed no magnetic attraction. In places the magnetite has been replaced by hematite along cracks and seams. There is slight surficial alteration of hematite to limonite also along seams, and within the masses of ore in places are minute cavities filled with soft, claylike material.

On the borders of the iron orebodies and also interbedded with them in places are banded quartzose ferruginous rocks, in places strongly schistose and containing microcline in addition to the usual quartz and iron oxide. These rocks resemble the Itabira iron-bearing siliceous rock of Minas Geraes, Brazil. They may have originated as sediments but have been much altered in places. Where weathered, as on the edges of steeply dipping beds, the iron oxide has partly altered to limonite and some of the silica has been dissolved out, leaving a porous, friable ore. Such porous material where encountered in drilling failed to yield a good core, but some of the sludge is rich in iron. A specimen of quartzose ferruginous rock from the north end of Gutierrez Hill was found to contain a trace of siderite, or iron carbonate, in the siliceous gangue. The siderite may be earlier than the magnetite, which occurs in small particles and veinlets following the direction of schistosity and breaking across the gangue minerals.

The streak of the hard ore is generally dark brown with reddish brown to grayish brown variations. This suggests an admixture of magnetite and hematite, or possibly the presence of oxidized magnetite.

Metallographic studies of polished sections of 11 typical specimens of the Pao iron ore, all except one from the outcrop, and all but one from

the main Boccardo area, including some of the quartzose, ferruginous schistose rock, were made by Dr. M. N. Short. Of the iron oxides, hematite predominates, but there is more or less magnetite present, which may be distinguished by its slightly brownish color in the polished section. The specularite ranged in size from grains averaging about 0.3 mm. dia. to coarse crystalline material of which some grains exceed 1 cm. dia. Figs. 14, 15 and 16 illustrate certain phases of these relations. Descriptions of these specimens follow, the metallographic notes having been furnished by Dr. Short (second paragraph under each number):

Specimen No. 1 is a medium-grained, dense, crystalline ore with banded structure in which the magnetite may easily be distinguished by the unaided eye. Magnetite and hematite are about equal in quantity. Strong magnetic properties.

Section consists of magnetite and hematite, the former predominating. Hematite replaces magnetite along cracks. The hematite over considerable areas has the same optical orientation. This may be due to pressure during regional metamorphism or it may be due to the magnetite being originally all part of one crystal and the replacement following a crystallographic direction in that crystal (Fig. 14).

No. 2.—A fine-grained, dense dark gray specular hematite showing faint banded structure. Magnetic.

Section shows specularite in interlocking grains averaging about 0.3 mm. diameter.

No. 3.—Medium-grained, dense, crystalline ore showing banding produced by weathering. Magnetic in hand specimen.

Section consists of crystalline hematite. Magnetite is absent.

No. 9.—Similar to No. 1 but with some platy hematite. Magnetic.

Section practically identical with No. 1. Hematite and magnetite are present in approximately equal amounts. Boundaries between the two show that hematite is later than magnetite.

No. 10.—Medium-grained, dense crystalline ore with considerable platy hematite. Magnetic.

Section shows mostly coarse-grained twinned hematite similar to No. 17. Also contains a few scattered grains of magnetite. The boundaries between the two are identical in character with those described under No. 1. Magnetite is undoubtedly the earlier of the two minerals and is being replaced by hematite.

No. 11.—Drill core from hole 11, consisting of fine to medium-grained ore with minute cavities containing white, claylike powder. Nonmagnetic.

Section of drill core consists of coarse-grained hematite, much of it twinned. Magnetite is absent.

No. 17.—Coarsely crystalline ore, slightly magnetic. Coarseness is exceeded in one other specimen in the collection.

Section consists of coarse crystalline hematite. The rhombohedral twinning directions can be seen on the polished surface without the aid of a microscope. The size of some grains exceeds 1 cm. dia. Such large grains can result from either regional metamorphism or magmatic deposition (Fig. 15).

No. 23.—Banded siliceous gray rock containing iron mineral in finely disseminated grains. Feebly magnetic. Tests show presence of iron carbonate. From north end Gutierrez Hill.

Section shows magnetite in small veinlets for the most part following direction of schistosity. Some, however, break across the gangue minerals. Hematite is absent.

No. 25.—Banded rock consisting of quartz and iron mineral more or less altered to limonite. Feebly magnetic.

Section very similar to No. 48 but hematite grains are much thicker in No. 25.

No. 36.—Small mass of bladed crystals of corundum containing small proportion of iron mineral. Nonmagnetic. (See Fig. 17 of thin section.)

Section contains coarse crystalline hematite. Differs from No. 17 in that the crystals are untwinned.

No. 48.—Banded quartzose ferruginous rock in which the iron mineral appears as fine gray to black grains and seams. Nonmagnetic.

Section shows hematite following pressure planes in schist. This is shown in detail in Fig. 16.

The specific gravity was determined upon six specimens of typical hard crystalline ore, ranging from medium grained to very coarse grained. The lowest specific gravity, 4.665, was shown by the coarsest grained material, which probably was more porous than the other specimens, and the highest, 5.140, by a dense, wholly crystalline material that shows perceptible magnetic properties. The average for the six specimens was approximately 5. With a specific gravity of 5, a solid mass of iron ore would weigh 1 gross ton to every 7.17 cu. ft. In calculating tonnage, however, it must be taken into consideration that at the surface this ore is badly jointed and contains many crevices which extend to some depth, as shown by drilling, so that a more conservative figure should be taken for the assumed volume per ton.

Chemical Composition

Chemical analyses of surface samples of the ore as well as of diamond drill cores disclose an iron ore of great purity throughout the greater part of the Pao deposits. The analyses show an iron content generally of from 65 to 70 per cent., although in some instances, due to the presence of magnetite in the ore, more than 70 per cent. of iron may be present. The content of phosphorus is generally below the Bessemer limit and that

of other deleterious elements such as titanium and sulfur are likewise low. Where the content of iron is high the silica and phosphorus are low but on the borders of the deposits silica and phosphorus usually increase.

The sampling of the surface exposures was done with great care and thoroughness under the direction of F. D. Pagliuchi and the analyses of these samples were made by Ledoux & Co. of New York. The records of a few are given in Tables 1 and 2.

TABLE 1.—*Analyses of Surface Samples of Iron Ore from Pao Deposits*
(Percentages after Drying.)

No.	Fe	SiO_2	Cu	Mn	Ni	Cr	Al_2O_3	As	TiO_2	S	P
A........	68.00	0.36	None	0.27	None	None	0.09	None	0.12	0 01	0.051
B........	69.25	0.38	None	0.09	None	None	0.33		0.08	0.02	0.028
C........	69.05	0.46	None	0.08	None	0.08	0.28	None	0.20	0.027	Trace (less than 0.01)

A Composite of 13 samples taken on Boccardo Hill over a distance of 1400 ft.; average width of ore, 76 ft.

B. Composite of 6 samples taken on Boccardo Hill over a distance of 800 ft.; average width of ore, 360 ft.

C. Composite of 5 samples taken on Picacho Hill over a distance of 1000 ft.; average width of ore, 161 ft.

Thirty samples of ore, 21 of them taken on Boccardo Hill, 5 on Picacho Hill, and the remainder from intermediate hills, and including some of the siliceous border ores, showed a range in iron from 62.5 per cent. to 69.8 per cent., in manganese from 0.06 per cent. to 2.92 per cent., and in phosphorus from a trace (less than 0.01 per cent.) to 0.22 per cent., and the unweighted average of this series of analyses is iron 67.64 per cent., manganese 0.25 per cent., and phosphorus 0.0355 per cent.

TABLE 2.—*Analyses of Iron Ore from Diamond-drill Core No.* 1

Depth, Feet	Fe, Per Cent.	Mn, Per Cent.	P, Per Cent.	S, Per Cent.	SiO_2, Per Cent.	Al_2O_3, Per Cent.
35	71.33	0.03	0.020	0.028	0.09	1.97
50	68.14	0.03	0.024	0.029	0.16	3.29
100	68.75	0.03	0.020	0.112	0.09	2.65
150	67.81	0.05	0.030	0.118	0.05	4.04
200	69.74	0.03	0.023	0.099	0.13	1.01
250	70.73	0.12	0.028	0.067	0.10	1.03
300	68.86	0.02	0.013	0.048		
350	68.53	0.05	0.025	0.039		
400	68.20	0.04		0.08	0.56	1.30
420	67.52	0.03		0.01	0.64	3.17

Total depth, 439.

Analyses of solid portions of the iron ore from drill cores show even better results than those obtained by sampling the outcrops. The analyses in Table 2, of the core from the first drill hole, made in the laboratory of a large steel plant, are typical. Other drill cores show ore of high quality but less thickness.

Suggestions as to Origin

The character and field relations of the Pao iron ores lend support to the possibility that either of two quite different modes of origin may be ascribed to the ores; for instance, according to whether the iron oxides were laid down originally as chemical precipitates in beds of sandstone and subsequently metamorphosed or the bulk of the iron oxides was introduced in heated solutions given off from molten rocks that invaded and intruded the overlying series of sediments.

The association of the Pao iron ore with banded quartzose rock into which it grades in many places and the resemblance of this quartzose rock to the Itabira iron formation, popularly called itabirite, a finely laminated rock composed of an intimate mixture of quartz sand and iron oxide that contains the iron ore in Minas Geraes, Brazil, and the relations of the ore in both fields to an underlying basement complex of granitic rock, as well as the high purity of the hematite within the orebodies in the two fields point to a similarity in origin of the Pao and Minas Geraes ores. Harder[2] and associates have brought forth much evidence to show that the Minas Geraes ores may have originated as sedimentary ferruginous beds, the richer portions, composed of specular hematite and a little magnetite, having been produced by regional metamorphism, and their views have been widely accepted.

On the other hand, study of thin and polished sections of the Pao iron ore, together with the ore itself in hand specimens, large masses, and from photographs affords many reasons for the belief that the richer parts, at least, of the orebodies may be of magmatic origin. For instance, petrographic studies were made by Dr. C. P. Berkey[3] in 1926 of a series of rocks and ores from the Pao deposits. The conclusion drawn by Dr. Berkey from these examinations was that the iron ore is of igneous origin, occurring as a veinlike development accompanying an intrusive rock, and having the essential features of a vein-dike. Resemblances of the streaked granular rock composed of quartz and iron mineral to a meta-

[2] E. C. Harder: The "Itabirite" Iron Ores of Brazil. *Econ. Geol.* (1914) **9**, 101.

E. C. Harder and R. T. Chamberlin: The Geology of Central Minas Geraes, Brazil. *Jnl. Geol.* (1915) **23**, 341, 385.

E. C. Harder: The Iron Industry in Brazil. *Trans.* A. I. M. E. (1915) **50**, 143–160.

C. K. Leith and E. C. Harder: Hematite Ores of Brazil and a Comparison with Hematite Ores of Lake Superior. *Econ. Geol.* (1911) **6**, 670.

[3] Private report to owners of property.

morphic rock were noted but it was believed that the deformation in it was due to movements in the magma during cooling and crystallization rather than to metamorphism. Had Dr. Berkey seen the relations of these rocks in the field, however, he would doubtless have been impressed with the gradation from quartzose ferruginous rock strongly resembling sedimentary beds on the one hand to the schistose quartzose ferruginous beds associated with the granite on the other hand.

The presence of corundum crystals within the hematite, indicating high-temperature conditions, is of significance, as well as the fact that the corundum crystals appear to be earlier than the hematite. The relations of the hematite and magnetite in this connection are also of much interest. In the polished sections Dr. M. N. Short finds instances where magnetite appears to have been the first of the two iron oxides to crystallize. He finds hematite to have replaced magnetite along cracks, but some of these minor alterations from magnetite to hematite may be supergene, as the material so examined came from the surface. Since most of the hematite is beautifully crystalline material, it seems hardly possible that the great masses of this mineral that make up the bulk of these deposits can be the products of supergene alteration from magnetite.

Several points that seem to have a bearing on the problem of the Pao ores have been brought out by Gilbert.[4] Among them are his conclusions that platy hematite is almost invariably of hypogene origin, that of the hematite which replaces magnetite the greater part is hypogene, and that only rarely does true hematite form as a product of surface oxidation of magnetite. He points out that replacement of magnetite by hematite is most vigorous in ores that are poor in sulfur and, as mentioned above, the Pao ore is low in sulfur. Magnetite exposed to surface oxidation is rather inert, and so it seems more reasonable to believe that, if magnetite is the primary mineral, the hematite has been formed by alteration of the magnetite at high temperature by the ore solutions themselves.

On the other hand, there is the possibility that the two minerals were formed simultaneously and that the proportions of hematite and magnetite were determined by the quantity of ferrous iron originally available. Broderick's[5] studies of the relations of the two minerals in iron ores from Juragua, Cuba, and from Sweden indicate that hypogene magnetite and hematite were probably formed almost simultaneously, but that later some of the magnetite was replaced by hematite of a second generation. These relations seem to fit the conditions of the purer central parts of the Pao ores most reasonably.

It has been the privilege of the writer to see several of the iron-ore deposits in Minas Geraes, Brazil, and thus to make direct comparison of

[4] G. Gilbert: Some Magnetite-hematite relations. *Econ. Geol.* (1925) **20,** 587.

[5] T. M. Broderick: Some of the Relations of Magnetite and Hematite. *Econ. Geol.* (1919) **14,** 353.

the ores from the two fields. One variety of the Pao ore closely resembles the typical Minas Geraes ore in the collection of the writer. This is a fine-grained specimen composed of specularite in interlocking grains averaging, according to Dr. Short, about 0.3 mm. dia. Most of the Pao ore is denser and more solidly crystalline than the ore from Minas Geraes, and moreover, the quartz in the quartzose ferruginous rock at Pao is not a granular sand as in the Itabira iron formation but has been largely recrystallized, with the hematite following the direction of schistosity and occupying cracks in the quartz. Notwithstanding these slight differences, which may be due to differences in degree of metamorphism, it seems possible that in the Pao area the siliceous ferruginous rocks on the borders of the rich iron orebodies, and interbedded with them in places, may be of sedimentary origin. These quartzose beds at El Pao, therefore, may be remnants of sedimentary deposits that formerly were more widespread but that still are represented to a greater areal extent and thickness in other parts of the Guayanan Highlands. According to Liddle[6] the age of the ferruginous quartzites of the Imataca series is not definite but is probably early Paleozoic and is therefore much later than the granitic basement rock of the Guayana series. The granitic and gabbroic rocks associated with the ore at El Pao may not, however, be the basement complex of Liddle but may be later than the quartzose ferruginous rocks. These quartzose beds at El Pao have been metamorphosed to schist in places and the richer parts of the iron ore have been profoundly affected by metamorphism, or else they may be in large part the result of replacement of the quartzose ferruginous beds by magmatic solutions. There are faint bandings in places in the richer ore which suggest that replacement may have played a part in the process. The metamorphosed ferruginous beds have had, therefore, a history somewhat similar to that involved in the formation of the iron ores of Minas Geraes, Brazil.

It seems most probable to the writer that the iron ores at El Pao consist of sedimentary quartzose ferruginous rocks, which have been more or less metamorphosed and which have received great enrichments from iron-bearing magmatic solutions that permeated the beds during their metamorphism. The source of the iron-bearing solutions may have been the granitic rock exposed next below the deposits of ore, unless it should be shown that the granitic rocks are older than the quartzose beds. There are also gabbroic rocks that may be suspected as well as the granite. The time spent at the deposits was altogether inadequate to establish these relations at all definitely.

The question as to the origin of a deposit of iron ore may or may not have a practical significance. The largest deposits of high-grade iron ore at present known are probably those in Minas Geraes, Brazil, where some

[6] R. A. Liddle: *Op. cit.*, 54, Fig. 3.

individual deposits are considered to contain between 300 million and 400 million tons of ore. It is claimed by engineers who have studied them thoroughly that the chances for continuity in thickness and quality are greatly enhanced because the ore originated as sedimentary beds. The tonnage of the great bedded nonmetamorphosed deposits of iron ore in Alabama, Newfoundland and Alsace-Lorraine can be estimated far more dependably in advance of drilling because of their sedimentary character than can, for instance, the lenticular magnetite deposits of the Adirondacks or of the Southern Appalachians, which are of magmatic origin. On the other hand, there are, perhaps, adherents to the belief that ore of purely magmatic origin may be richer and more deeply seated, and consequently more extensive than if it originated otherwise. It would seem, therefore, to be an advantage for a deposit of iron ore to have both sedimentary and igneous antecedents.

The richness of the Pao ore in all probability is related to metamorphic and magmatic action, but the depth will be limited of course, by the depth of the igneous source of the iron. The igneous rocks outcrop 200 to 500 feet lower than the orebodies which have not been found by drilling at greater depths and can hardly be expected to continue much deeper unless they have been folded and faulted into the igneous rocks. The present deposits are apparently the roots of formerly much more extensive deposits, the upper parts of which have been removed by erosion, and they represent a combination of processes of origin that has resulted in the formation of large and rich bodies of iron ore.

Other Deposits of Iron Ore in Orinoco River Region

Six deposits of iron ore are known in the region south of Rio Orinoco east of the Pao deposits. According to published information,[7] they occur in a narrow belt in the northern part of the Sierra Imataca in the highly ferruginous quartzites of the Imata series, possibly faulted or folded into the old gneissoid basement complex. It is stated that although practically the entire range is formed of grayish black, iron-bearing quartzite having a metallic luster, the richest deposits occur near igneous intrusions, which is a point of similarity to the Pao deposits. Locally these deposits have been found at Los Castillos, Piacoa, Santa Catalina, Sacuroco, Manoa (Imataca), and La Escondida (Fig. 1) and have been prospected more or less, but only at Manoa (Imataca) and Piacoa have they been developed to any extent.

The Imataca mine is in the foothills of the Sierra Imataca near Caño Corosimo, 85 km. above Boca Grande of the Orinoco and about 140

[7] R. A. Liddle: *Op. cit.*, 385–387.
B. L. Miller and J. T. Singewald, Jr.: The Mineral Deposits of South America, 534–538, New York, 1919. McGraw-Hill Book Co.

ft. higher than the river. Iron ore has been found chiefly in two east-west veins, one of them about 8 ft. wide, which dip 40° to 80° south. The ore in these veins, chiefly hematite with a little magnetite, ranges generally from 65 to 70 per cent. of iron with low silica, phosphorus and sulfur, but the unenriched ferruginous quartzite carries much less iron. The gneissoid floor is traversed by quartzite and veins of hematite, indicating faulting. The mine exported to the United States 12,100 tons of ore in 1912 and 56,975 tons in 1913, but there is no record of later shipments. A summary of operations which apparently began about 1893 is given in the *Iron Trade Review.*[8]

The Piacoa, or Colon, mines, about 60 miles farther up the Orinoco River, are in a hill composed of highly ferruginous quartzite, 450 ft. high. The ore deposits are thin, small veins of dense, well-crystallized hematite, which has a grayish black luster on fresh surfaces but weathers brick red to reddish brown. A conglomerate, containing fragments from the ore veins and from the ferruginous quartzite, mixed with hydrated iron ore, has been formed, similar to the canga ores of Brazil. Average analyses of the Piacoa ore show a good grade of ore which contains a little more phosphorus than the other ores in this region. Locally the ferruginous quartzite is distinctly schistose, as it is in the two hills at Los Castillos and at other places, another resemblance to conditions at places in the Pao locality.

TABLE 3.—*Analyses of Iron Ore from Deposits South of Orinoco River in Venezuela*[a]

	Fe, Per Cent.	SiO_2, Per Cent.	Mn, Per Cent.	P, Per Cent.	S, Per Cent.	TiO_2, Per Cent.
Imataca, average 3 cargoes	66.53	1.81		0.031	0.045	0.139
Piacoa, average 6 samples	59.37	7.25	0.66	0.192	0.072	0.84
Piacoa, average	61.00	3.85	8.50	1.260	0.037	0.60
Los Castillos	59.00	6.50	0.10	0.112	0.052	0.90
Santa Catalina	64.00	1.20	0.07	0.140	0.022	0.30

[a] From R. A. Liddle and B. L. Miller and J. T. Singewald, Jr.: *Op. cit.*

Transportation difficulties, involving the transshipment of ore, are believed to have hindered the operation of these mines. At Imataca only boats carrying 2000 to 3000 tons can enter Caño Corosimo, and besides, the lower part of Boca Grande of the Orinoco contains so many sand bars that there was only sufficient water for limited transportation of the ore.

[8] Status of Venezuelan Iron Ore Development. *Iron Trade Rev.* (1913) **52**, 685.

Economic Considerations

Grade of the Iron Ore

Under the heading Chemical Composition, it has been shown that the Pao deposits are composed principally of an iron ore of remarkable purity. Many of the analyses show an iron content in excess of 70 per cent., the theoretical maximum for hematite, and this excess is accounted for by the presence of considerable magnetite, which contains 72.4 per cent. of iron, where pure. There is no doubt that the ore in the workable parts of the deposits is suitable for making Bessemer steel, and it should also be found suitable for smelting to pig iron and steel by electricity. Electric smelting requires an iron ore of high grade and magnetite generally has been supposed to be required for this purpose on account of its slightly higher content of iron than hematite. The ore, although predominantly of hematite, carries a higher percentage of iron than most commercial magnetite and possesses the additional advantage of carrying less titanium oxide (TiO_2) than ore that is chiefly magnetite. Moreover, it is superior in quality to some of the ores already being supplied to electric furnaces.

In view of the adaptability of the Pao iron ore for smelting by electricity and the proximity of large undeveloped water powers in the falls of Rio Caroni, also of the existence in the Orinoco delta region of extensive forests from which charcoal may be made, the interesting possibility of the manufacture locally of iron and steel by means of electricity comes to mind. However, the questions as to the feasibility of such a project, and other economic problems aside from the supplies of raw materials involved, are beyond the scope of this paper.

Areal Extent of Ore

In the main area at El Pao, which includes Boccardo, O'Callahan, and Picacho hills, Loring Knob, North Knob and the strips of outcropping iron ore which practically connect these larger bodies in an elliptical figure, a linear extent of about 6335 ft. of almost continuous iron ore may be measured. The width of visible ore around this elliptical strip ranges from 40 to 400 ft. and the sum of the various unit areas, considered in making estimates of tonnage, aggregate about 15 acres. There is, in addition, a probable area of 5 acres in the forested basin between Boccardo and Picacho hills that is, according to drill records, underlain by iron ore concealed by surface material, clay, canga, gabbro and serpentine. On Gutierrez hill there is indicated an area of about 7.6 acres of ore outcrop. There are many small scattered outcrops of iron ore in the vicinity of the larger areas, a general estimate for which gives a total area of about 1.35 acres. Generally where surface ore is visible there is much loose talus of ore but no rock of any other kind is

exposed and trenching to depths of several feet in places in the loose ore has failed to reach the bottom of it, so that it is difficult to distinguish between float ore and ore in place. In addition to the areas in which there is good evidence for the presence of iron ore, there is east and southeast of the Upper Camp a large, nearly flat forested tract of about 33 acres. This area is surfaced with reddish clay containing much limonite and hematite in fine shot and gravel sizes. It has not been prospected except by a few test pits, some of which disclosed boulders of iron ore in clay at a depth of a few feet. The fact that this surface has remained at nearly the same level as that of the exposed iron ore capping indicates that a hard material such as iron ore may have protected the surface from erosion into gullies. Moreover, around the border of this area there are cliffs of iron ore in many places. There is thus indicated a grand total of more than 28 acres probably underlain by iron ore besides the Upper Camp flat where there is a possibility of ore beneath a still greater area.

Prospecting

The quantity of iron ore visible at El Pao is impressive, particularly because of the large area covered by almost pure, bright metallic-appearing hematite and magnetite and because there is no other rock in sight in the upper parts of the orebodies. Since there is necessarily a large quantity of unconsolidated ore in large slabs and blocks on the tops and slopes of the hills derived through disintegration of former higher portions of the orebodies, the actual extent of the ore in place can be determined only by thorough test-pitting, trenching, core drilling and tunneling. The unconsolidated ore is so deep and the blocks are so heavy on Boccardo and Picacho hills that a trench several feet deep crossing both of these hills is not conclusive in several places as to the actual subsurface conditions. The topographic expression should be some guide to the presence of iron ore, as the high-grade ore is much harder than the granodiorite basement rock and the quartzose ferruginous rock that borders the deposit, so that ridges or flat-topped areas are quite likely to be due to the presence of ore. No tunneling has been done but this method of prospecting, although arduous, would probably prove more feasible than horizontal drilling and would reveal some essential information if tunnels could be driven into some of the hillsides.

A campaign carried on in 1928–29 completed a total of more than 5000 ft. of diamond-bit core drilling, 14 holes having been put down ranging from 43 to 515 ft. in depth. Nine of the holes were vertical but two were drilled at an angle of 45° and three at 60° to the horizontal. The locations of these holes are shown on the map, Fig. 2. The holes that went through good ore into quartzose ferruginous rock showed orebodies ranging in depth from 16 to 400 ft., having in some places alternat-

ing layers of good ore and quartzose ore. Some holes that reached the top of the orebody were discontinued in the ore. A light, gasoline-driven equipment was used, apparently inadequate for drilling such refractory material as the hard iron ore proved to be. The damage to diamonds was great and two bits were lost beyond recovery. Crevices in the iron ore and the alternation of hard and soft material added to the difficulties of drilling. More drilling should have been done in order to satisfactorily prospect the deposits, but when the contract was fulfilled conditions were not favorable to continuation of this work. Additional holes nearer to Picacho Hill are needed in order to determine the thickness of the ore in that locality, and holes at several other places would have been desirable, particularly on the west slope of Boccardo Hill and in the flat area south and east of the Upper Camp. A survey with a magnetometer would perhaps be helpful in the latter place and in basins thought to contain ore.

Sections *A-A′*, *B-B′* and *C-C′*, Fig. 2, indicate in a generalized way the profiles of the several drill holes, but the data are not sufficient to enable satisfactory correlations to be made.

Estimates of Ore Tonnage

In view of the limitations outlined, the most carefully made estimates of tonnage will be far from accurate. The least definite of the dimensions of the orebodies is that of depth, and favorable interpretations must be made of some of the drill data in order to use them in this connection. The most successful drill holes, considered together with the cliffs of solid hematite west of Boccardo Hill, on North Knob, southeast of the Upper Camp and on Gutierrez Hill, indicate that much of the ore extends to as great depths as would be convenient to reach in open-pit mining. The figures used in making the estimates herewith are not entirely warranted by the data at hand, but seem as reasonable as can be derived for each unit area, therefore the tonnages should exist in proportion to the degree in which certain assumed dimensions prove to be correct. In calculating tonnage it was necessary to take into consideration the fact that the ore in place at the surface is badly jointed and contains many crevices which extend to some depth, as shown by drilling and also by the fact that there is much float ore on the surface that cannot be regarded as solid ore. According to the specific gravity of the solid ore, 8.5 cu. ft. might conservatively have been considered as equivalent to a long ton, but in order to incline well beyond the limit of safety, a factor of 10 cu. ft. per long ton was used.

In April, 1926, when less was known of the Pao deposits than at present, William J. Millard estimated that there were about 12,000,000 tons of iron ore in sight and about 18,000,000 tons of probable ore, a total of 30,000,000 tons. The totals arrived at by the present writer exceed those of Millard slightly; the quantity of ore estimated as in sight and as

probable exceeds 35,000,000 tons, and may be divided in the proportion of about 15,000,000 tons in sight and about 20,000,000 tons probable. A large additional quantity of ore may be considered as possibly underlying the nearly flat area in the vicinity of the Upper Camp, as indicated by the border of iron ore that practically surrounds this area. Nothing definite will be known, however, concerning this possibility until it is determined by drilling or magnetic prospecting.

The order of magnitude of the Pao iron-ore deposits is obviously greater than that of the hematite and magnetite deposits on the south coast of Cuba, such as the Daiquiri, Juragua, El Cuero, and Guamá, which have to date shipped about 21,000,000 tons of ore, and which are understood to be largely exhausted.

Mining

The deposits of iron ore at El Pao are high above the local drainage level, so that mining either from open cuts or underground will not be hindered by ground water. The large surface exposures can certainly be mined best from open cuts to whatever depth seems practicable, and work may be continued underground. Mining hard hematite underground has long been carried on successfully in northern Michigan and Wisconsin and costs have been reduced to very reasonable figures in recent practice.[9] Underground iron mining has also been carried on for more than 80 years in the magnetite deposits of the Adirondack region in New York State, and this mining practice has recently been described.[10] With these two prominent examples, and many others that could be cited, of successful mining of iron ore underground, most of them in deposits of lesser dimensions than those at El Pao, there is ample assurance that these deposits can be mined advantageously at depth as well as from open cuts.

In a tropical country like Venezuela, in the lowland the temperature is undoubtedly high during much of the year, but at the locality of the Pao deposits if men are protected from the sun's rays the breeze makes the temperature very agreeable, while the nights, even in midsummer, are refreshingly cool, necessitating the use of a woolen blanket while sleeping.

According to the drill records there are iron orebodies in the basin between Boccardo and Picacho hills that are covered by material that would be almost too thick to be stripped off in open-cut mining, whereas in underground mining the winning of the ore would be a simple matter. There is plenty of timber in this part of Venezuela, and if the water power at the falls of Rio Caroni is developed electricity can be delivered

[9] L. Eaton: Method and Cost of Mining Hard Specular Hematite on the Marquette Range, Michigan. U. S. Bur. Mines, *Information Circular* 6138 (1929). Also *Eng. & Min. Jnl.* (1929) **127**, 991.

[10] A. M. Cummings: Method and Cost of Mining Magnetite in the Mineville District, New York. U. S. Bur. Mines, *Information Circular* 6092 (1928). Also *Eng. & Min. Jnl.* (1929) **127**, 190, 234.

at El Pao for operating mining machinery, lighting the mines and village, pumping water to the mines, and for operating the railroad for carrying ore to Rio Orinoco.

Markets

The principal nations that consume iron ore are the United States, Germany, France, United Kingdom, Belgium, Italy, Spain, and Japan. Of these countries the United States possesses the largest developed reserve of iron ore within its own boundaries, but as iron ore is a mineral product of comparatively low value, and has to be transported in large quantities, it has proved cheaper to import foreign ore at Atlantic seaboard ports than to ship it from interior fields. This practice will probably grow and may eventually include ports on the Gulf of Mexico. The imported ore has come from Cuba, Chile, Spain, northern Africa and Sweden. The United States may, however, be considered self-sufficient so far as supplies of iron ore are concerned.

Germany, since the loss of the Lorraine iron ore fields, has depended largely upon imports of iron ore from Sweden and Spain, and cannot under present conditions be considered self-sufficient in this important ore.

France is well supplied with iron ore and probably can at present be considered in a position to export the ore.

The United Kingdom has considerable low-grade iron ore within its borders, but the British iron and steel industry requires considerable high-grade ore in addition to the domestic production and is therefore an importer of ore.

Belgium is not well supplied with high-grade ores of iron and is obliged to depend largely upon imported ore.

Italy is also dependent upon foreign deposits of iron ore to supply domestic iron manufacturers and, moreover, does not manufacture sufficient iron and steel to supply her domestic requirements.

Spain is more than self-sufficient in reserves of iron ore, having long been an exporter of this product.

Japan does not possess sufficient iron-ore reserves to meet domestic demands and is not known to have large or easily accessible supplies of ore available elsewhere in the Orient. This country, therefore, although remote from Venezuela, might be a potential market for ore if it were shipped through the Panama Canal and across the Pacific.

It is therefore apparent that of the eight nations mentioned the six largest consumers are importers of more or less iron ore, and that certain of them, such as Italy, Germany, Belgium, and the United Kingdom may well be considered as potential markets for high-grade Venezuelan iron ore.[11]

[11] Statistical data of great interest in this connection have been compiled in Mineral Raw Materials, *Trade Promotion Series* No. 76, by J. W. Furness and others. U S. Department of Commerce (1929).

DISCUSSION

(R. J. Colony presiding)

E. C. Harder, New York, N. Y. (written discussion).—In reading Mr. Burchard's paper and in talking with Mr. Burchard and with others familiar with the Venezuelan iron ores, I have been struck by the apparent similarity of these ores to the iron ores of Minas Geraes, Brazil. As is well known, the distinctive feature of the Brazilian iron ores is their association with laminated ferruginous quartzite, called itabirite—in which they commonly occur as beds or lenses conformable with the lamination. The iron-bearing quartzite is part of a regular sedimentary series, being underlain by sericitized quartzite in places of great thickness and overlain by schist containing local limestone lenses. The only rock of igneous origin that has been found within the sedimentary series in Minas Geraes is serpentine. This occurs in only two or three places as small lenses or irregular bodies and is not associated with the iron ore or itabirite but is found in the sediments above and below. Whether it is intrusive or extrusive is undetermined. Granite and gneiss are the predominant rocks on which the sediments rest. No granite intrusive into the sedimentary series has been found, therefore it is generally agreed that probably the Brazilian iron ores are of sedimentary origin.

Not having seen the Venezuelan iron ores in the field, I am not qualified to make any definite statements regarding them, but judging from Mr. Burchard's description, and also from a recent statement prepared by Mr. Zuloaga,[12] the general similarity of these ores to those in Brazil is striking. There is in Venezuela, as in Brazil, a laminated ferruginous quartzite in which the iron ore apparently is found as lenses in the main conformable with the laminations. The sedimentary series of which the ferruginous quartzite forms a part is underlain in both places by a basement complex of granite and gneiss. No granite intrusives have been found definitely within the sedimentary beds. Serpentine and gabbro, however, locally occur with the sediments. Mr. Burchard and Mr. Zuloaga are inclined to believe that possibly some of the granitic rocks of the supposed basement complex in reality may be intrusives of later age than the sediments, but proof of this is lacking.

Both Mr. Burchard and Mr. Zuloaga, although admitting the association of the iron-ore deposits with undoubted sedimentary iron-bearing rocks, are inclined to look to magmatic sources for at least a part of the iron in the deposits themselves. This would seem unnecessary; for having abundant iron available in the ferruginous quartzite we may assume transfer and redisposition through metamorphic agencies with resulting concentration, without calling upon iron from outside sources.

The Venezuelan iron ores lack the earmarks so abundantly displayed by most iron ores deposited from solutions of igneous origin. Granitic or dioritic intrusives such as are almost invariably associated with these iron ores have not been shown to be present near the Venezuelan iron ores. It is true that gabbro occurs as an intrusive, but generally iron ore that is associated with gabbro elsewhere in the world is titaniferous. There is no titanium in the Venezuelan ore. None of the common metamorphic minerals present in iron ores of igneous origin are found with the Venezuelan iron ores. The only minerals aside from iron minerals and quartz mentioned by Mr. Burchard are microcline and corundum locally present, while Mr. Zuloaga mentions finding garnet and veins of hedenbergite but does not say that they occur near any of the important iron ore deposits. Altogether, the iron ores seem to be characterized by a significant absence of complex metamorphic minerals and igneous relationships.

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[12] G. Zuloaga: The Iron Deposits of the Sierra de Imataca, Venezuela. *Econ. Geol.* (1930) **25**, 99–101.

Briefly, the Venezuelan iron ore deposits occur along a belt of sedimentary rocks stretching east and west for many miles along the south side of the Orinoco valley. Ferruginous quartzites are conspicuous in this belt and the iron ores are associated with them. In the absence of definite evidence such as igneous intrusions and complex metamorphic minerals, why should we suppose that solutions of igneous origin selected this long, narrow belt of difficultly replaceable iron-bearing sediments for the further introduction of iron, leaving the more readily replaceable marginal rocks unaltered? Having, in any case, a rich sedimentary iron-bearing formation, it would seem more reasonable to consider the iron-ore lenses either original sedimentary deposits or reconcentrations of original material through metamorphism.

M. Roesler, New York N. Y. (written discussion).—Mr. Burchard mentions the difficulties encountered in attempts to mine the Orinoco deposits. I believe he has not mentioned one of the greatest—the disease due to insufficient knowledge of the sanitary measures needed in tropical countries.

In the Cuban deposits the magnetite and hematite are undoubtedly of hypogene origin and syngenetic. They also offer strongly confirmatory evidence of the solid solution theory which was advanced by the Carnegie Geophysical Laboratory some time ago. So much so that I have rather abandoned the usual nomenclature in favor of magneto-hematites.

The dual origin, which Mr. Burchard suggests, and the replacement of magnetite by crystalline hematite are very suggestive. Would it be possible that there are two periods of mineralization of magmatic origin—the first of a magnetite and the second of a hematite with excess oxygen?

To form bodies of such an extent as Mr. Burchard shows by dynamic metamorphism would, I believe, require a more intense metamorphism than I can find in the picture Mr. Burchard draws. The origin that Dr. Berkey suggests from his study of the slides seems the more probable.

J. W. Gruner, Minneapolis, Minn. (written discussion).—The Pao deposits described by Mr. Burchard resemble the hard ores of the Soudan formation of the Vermilion range of Minnesota in some respects:

1. The orebodies consist of concentrations of hematite in relatively lean iron-bearing formations.
2. The original bedding of the iron-bearing formations has been almost obliterated in the ores.
3. Silica has been removed almost completely. In the Pao deposits this silica was in the form of quartz, on the Vermilion range it was fine-grained quartz and jasper.
4. The silica was largely replaced by specular hematite, which was brought in by probably hot solutions.
5. Magnetite is mostly oxidized to or replaced by hematite.
6. Dynamic metamorphism did not follow but preceded the formation of the orebodies. This is shown in the Soudan ores by their cutting across folds, also by their association with dikes which cut the highly folded iron formation. Neither shearing nor schistosity is preserved in the ores. Similar conditions seem to exist at Pao.

While basic intrusives are almost the rule in and near the ores of the Vermilion range, such intrusives have not been recognized in the Pao ores, though it is probable that the granodiorite or gabbro which have been found are intruded into the iron formation according to Mr. Burchard.

I have lately[13] advanced a hypothesis for the origin of the soft and hard ores of the Lake Superior region, which may also be applicable to the Pao deposits. Experi-

[13] J. W. Gruner: Hydrothermal Oxidation and Leaching Experiments; Their Bearing on the Origin of Lake Superior Hematite-limonite Ores. *Econ. Geol.* (1930) **25**, 837.

ments with hot water at temperatures between 150° and 300° C. have shown that ferrous iron minerals including magnetite are oxidized in the absence of air by these hot waters if the hydrogen liberated in the reaction can escape readily. Practically all iron is precipitated in this reaction while very large quantities of silica, including quartz, are taken into solution. Hot waters rich in Na, Mg, Ca and Fe, therefore, from intermediate or basic magmas could produce replacement deposits of the type of the Pao and Vermilion orebodies.

W. L. CUMINGS, Bethlehem, Pa. (written discussion).—We should not conclude too hastily that the igneous rocks at Pao had anything to do with ore origin. Under the tropical conditions no outcrops of the igneous rocks are seen near the orebodies, so actual relations cannot be determined. It might easily happen that the igneous rocks found in drilling are simply later dikes cutting the orebodies and have nothing to do with ore deposition. In the Lake Superior district, for instance, granite dikes are known to cut the iron formation in a few instances. If the actual relations there had been obscured by tropical weathering and the granite dikes only known by drill holes, it might have been thought that they played a part in ore deposition.

One type of igneous rock found in the drill holes at Pao was classified as serpentine. From this rock as taken from the drill a quantity of small black octahedral crystals was obtained by panning. Naturally, it was thought these might be chromite, on account of the association with a supposed peridotite alteration product. Analysis showed them to contain no chrome—they were undoubtedly crystals of martite.

W. H. NEWHOUSE, Cambridge, Mass. (written discussion).—During the past winter I visited a number of the occurrences of iron south of the Orinoco River, in company with Mr. Zuloaga. We were interested in the relations observed at the other occurrences in comparison with those seen at Pao. The well-banded, or bedded, itabirite, which we found in many places, is composed of alternating bands of quartz, and magnetite with hematite. It is undoubtedly a sedimentary rock very similar to the itabirite described from Minas Geraes, Brazil. At every place we visited where it was locally rich in iron it was near igneous rocks. At Monte Cristo, although a definite contact was not seen, about one hundred yards from the granite a strong contact metamorphic zone with garnet, sillimanite and biotite was found.

At Piacoa and El Becerro, veins of magnetite and hematite were found cutting across the bedding of the itabirite and at times connecting with small, rich, irregular masses of ore. Mr. Zuloaga, in his study of thin sections of the rocks, has found notable amounts of pyroxene along shear planes in the same rock.

These features and others[14] suggest some migration of the iron and suggest the possibility that that migration and consequent enrichment may be related to the intrusion of the igneous rocks.

E. F. BURCHARD (written discussion).—The discussion by E. C. Harder presents very strong arguments for the sedimentary origin of the Pao iron ore. Perhaps in the present state of knowledge concerning the deposits it may not be possible convincingly to refute them. It is, of course, conceivable that the richer parts of the iron deposits may have been derived from original sedimentary deposits and that they have undergone reconcentration of original material through intense metamorphism, which would place them in a class similar to the Brazilian deposits which they so greatly resemble. As stated by Mr. Harder, proof is lacking in the data gathered by the author that some of the granite is intrusive into the sediments, but drill records show that in at least four places gabbro is related to the

[14] G.. Zuloaga: *Op. cit.* A paper now being prepared will describe the relations in detail.

orebody as if cutting the beds at an angle to the marginal so-called sedimentary ferruginous beds, and, although the iron ore is nontitaniferous in the ordinary sense, it does contain fractional percentages of titanium dioxide comparable with those of the Adirondack nontitaniferous magnetites which, according to Kemp and Newland, probably have been introduced by processes connected with the intrusion of igneous masses, including gabbro. Despite the absence of direct evidence of igneous intrusions and of complex metamorphic minerals, the marginal quartzose ferruginous rocks actually have been altered to such an extent that Dr. Berkey considered them rather as parts of a magma, expressing the opinion that the schistose texture of this rock was due to movements in the magma rather than to metamorphism. That igneous emanations should have invaded the Sierra de Imataca is, of course, entirely accidental, and there is no particular reason why they should not have done so.

It is significant that of the additional opinions regarding the origin of the Pao ores, by Messrs. Roesler, Gruner, Cumings, Newhouse and Zuloaga, four out of five favor the magmatic hypothesis. Roesler's question as to the possibility of two periods of mineralization of magmatic origin received some consideration in reviewing the work of Gilbert and Broderick on iron ores, and the impression was gained that the bulk of the magnetite and hematite was formed at one time but that later some of the magnetite was replaced by hematite. If this later replacement is the second period of mineralization referred to by Roesler, such a process seems admissible.

Guillermo Zuloaga made some intensive laboratory studies on the problem at the Massachusetts Institute of Technology during the winter of 1929–30 and is expected soon to submit his thesis for publication. A new campaign of drilling the deposits was under way in the summer of 1930 with a geologist and a mining engineer on the ground.

As stated in the original paper, the opportunities for field study were altogether inadequate to establish the relations and history of the iron ore deposits at all satisfactorily. The author attempted to present as clear a description as was possible with the material available and merely to suggest the explanations that according to his judgment best fitted the facts. What the later and more thorough studies may bring forth will be awaited with interest.

World Production and Resources of Chromite*

By Lewis A. Smith,† Washington, D. C.

(New York Meeting, February, 1931)

Chromium is one of the new metals, but considerable research has been required to determine an approximate record of its production from 1827 until the present. Its use in the form of pure metal is not extensive; of the chromite mined, about one-half is required for making ferrochromium, about two-fifths goes directly into refractories and the remainder is absorbed chiefly by the chemical trades. The industries that demand chromite for such uses are mainly in countries that produce little chromite, while the important chromite-producing countries are not generally consumers. The international flow of chromite is therefore of interest, as well as the political and commercial control of resources of the metal.

World Production

The first mining of chromite is said to have occurred as early as 1820, at the Roros deposit in Norway, but the first well-established output came from the Reed mine, in Maryland, about 1827. Thereafter the United States led the world in chromite production until 1860, when Turkey took the lead. Russia in turn became the chief producer from 1897 to 1902, inclusive. From 1903 to 1909, New Caledonia held first place, but alternated with Southern Rhodesia from 1910 to 1917, inclusive. In 1918, 1919 and 1920–1921, the United States, India and New Caledonia led, respectively. In 1922 Southern Rhodesia assumed first place and has since continued to be the world's foremost producer by a large margin. Fig. 1 illustrates the production of the leading countries from 1890 to 1929, inclusive. Fig. 2 shows the world production of chromite from 1890 to 1929, inclusive, in long tons and Table 2 gives the production by countries for the same period.

The total world production for the 103 years, 1827 to 1929, inclusive, is estimated at about 7,100,000 long tons, distributed as shown in Table 1.

World Trade in 1928

In reviewing the figures for the flow of chromite in 1928, it is evident that the six countries that produce most consume little chromite and

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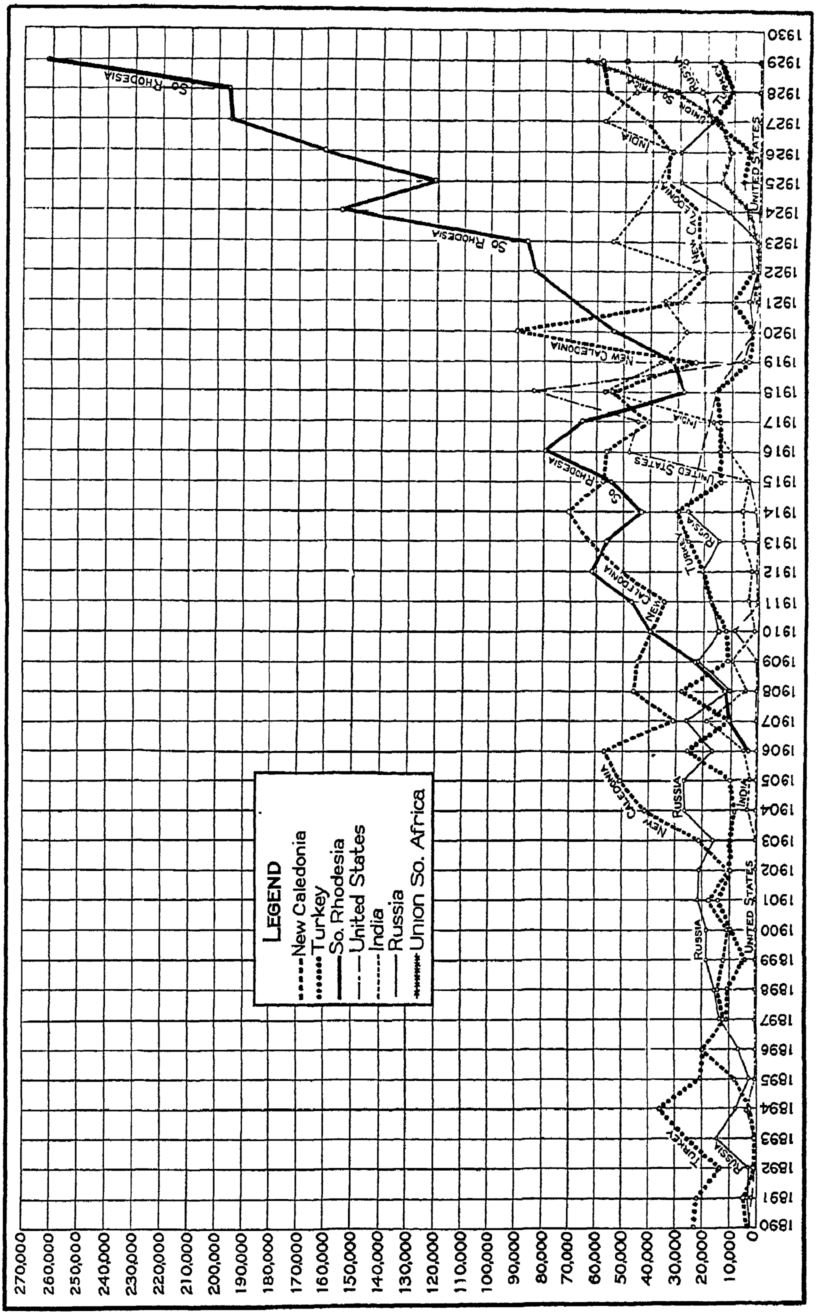

FIG. 1.—PRODUCTION OF CHROMITE IN PRINCIPAL PRODUCING COUNTRIES, 1890–1929 INCLUSIVE.

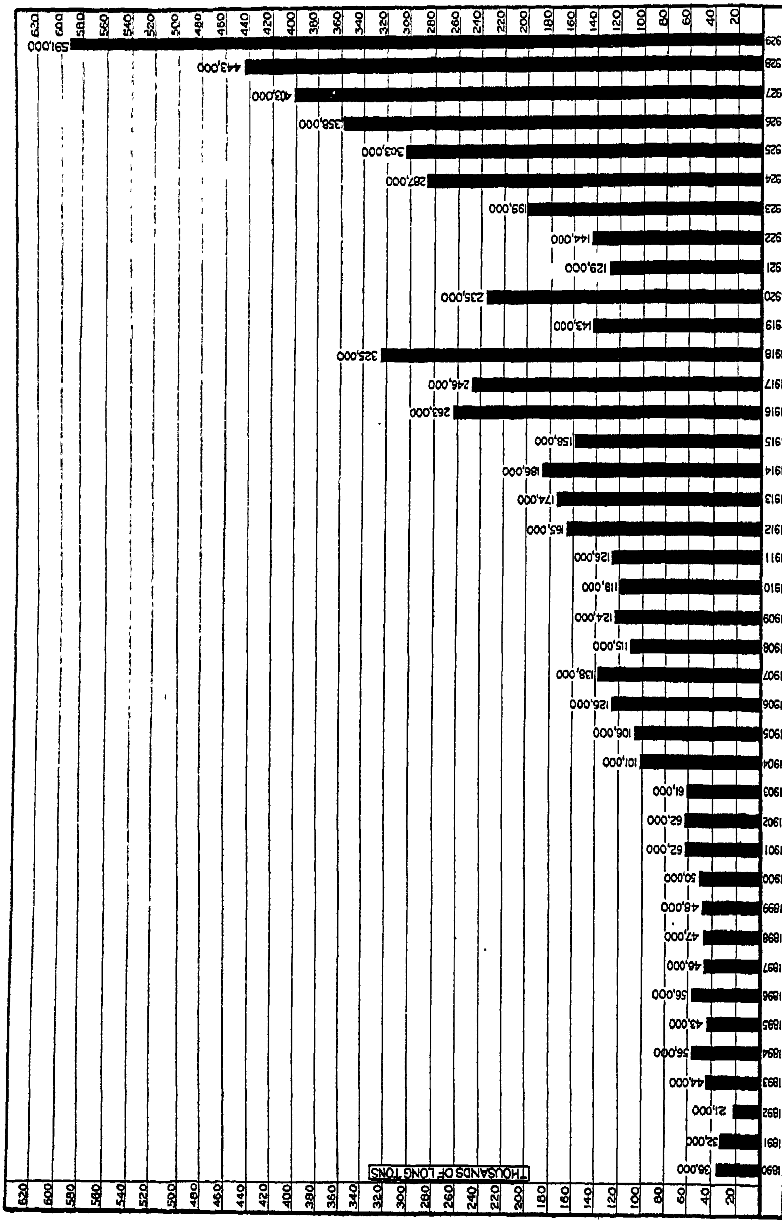

FIG. 2.—WORLD PRODUCTION OF CHROMITE IN LONG TONS, 1890–1929 INCLUSIVE.

that the countries that consume most, excepting Russia, produce only negligible amounts. These facts illustrate how an important manufacturing industry may depend largely upon imported raw materials.

The production of chromite in 1928 amounted to about 443,000 tons, of which nearly 87 per cent. was produced by Rhodesia, New Caledonia, British India, Cuba, the Union of South Africa and Russia. The world total exports amounted to about 364,000 long tons, 94 per cent. of which was exported by the countries mentioned. The total imports amounted

TABLE 1.—*World Production of Chromite*, 1827 *to* 1929, *Inclusive*

Country	Production, Long Tons[a]	Estimated Content, Per Cent.[a]	Per Cent. of Total
Southern Rhodesia	1,912,000	49–50	26.9
New Caledonia	1,346,000	50–52	19.0
Turkey	832,000[a]	40–51	11.7
Russia	650,000[a]	40–45	9.2
India	615,000	50–	8.7
United States	478,000	30–50[b]	6.7
Greece	336,000	38–45	4.7
Cuba	213,000	38–45	3.0
Canada	175,000	40	2.5
Union of South Africa	139,000	42	2.0
Japan	112,000	[c]	1.6
Others	282,000[a]	[c]	4.0
	7,091,000[a]		100.0

[a] Subject to revision.

[b] The bulk of the ore mined in the United States during the war ran from 30 to 45 per cent. (about one-fifth of the production contained 40 to 50 per cent.).

[c] Not available.

to around 350,000 tons, of which the United States, Germany, Sweden, the United Kingdom, Norway and France took 95 per cent. and consumed 87 per cent. of the chromite used during the year. Russia, Yugoslavia and Japan were the only nations that consumed large quantities that were able to produce sufficient chromite to meet their needs. Table 3 shows the amount of chromite produced, exported, imported and consumed by the principal countries in 1928. Fig. 3 shows the location of world chromite deposits, and the relative production, exports, imports and consumption of the principal producing and consuming countries during 1928.

TABLE 2.—*World Production of Crude Chromite, by Countries, 1920 to 1929, Inclusive*

LONG TONS

Country	1920	1921	1922	1923	1924	1925	1926	1927	1928	1929
Australia	1,617	62	529	1,192	773	963	597			129
Brazil[a]	3,451						1,476	1,791	20	69
Canada[b]	9,835	2,498	685	3,177						
Cuba[a]	710	600		10,420	19,567	29,830	36,020	16,984	33,984	52,950
Cyprus					2,811	1,989	516	700		829
Great Britain	1,100		595	546	1,043	448		378		
Greece	12,295	7,902	9,068	14,594	14,420	7,951	19,732	17,041	20,622	[e]
Guatemala[b]	1,122	401					53			
India	26,801	34,762	22,778	54,243	45,463	37,452	33,382	57,207	45,456	49,566
Indo-China					20					11
Japan	3,906	3,315	3,698	4,455	5,277	5,731	6,941	9,628	9,653	[e]
New Caledonia	90,089	28,993	19,068	22,853	22,657	34,184	33,726	42,208	56,004	58,213
Norway[a]				34	174	443	17			
Rhodesia (So.)	53,811	44,810	83,460	86,317	154,219	121,275	161,780	164,658	195,916	261,710
Rumania			30	59						
Russia[c]	2,918	3,950	834	878	11,706	29,636	29,883	18,989	21,620	27,560[d]
Turkey	24,605[d]	[e]	2,500[d]	[e]	3,346[d]	7,387	6,565	18,029	11,662	14,840
Union of South Africa		1,061	86		4,500	13,539	11,852	16,691	31,255	62,963
United States	2,502	282	355	227	288	108	141	201	660	269
Yugoslavia	10	10	15	[e]	295	11,968	15,731	8,619	16,415	42,343
Total	234,772	128,646	143,701	198,995	286,559	302,894	358,412	403,124	442,990	591,000[d]

[a] Exports. [b] Imports into United States. [c] Data for fiscal year ending Sept. 30. [d] Estimated on basis of incomplete data. [e] Data unavailable.

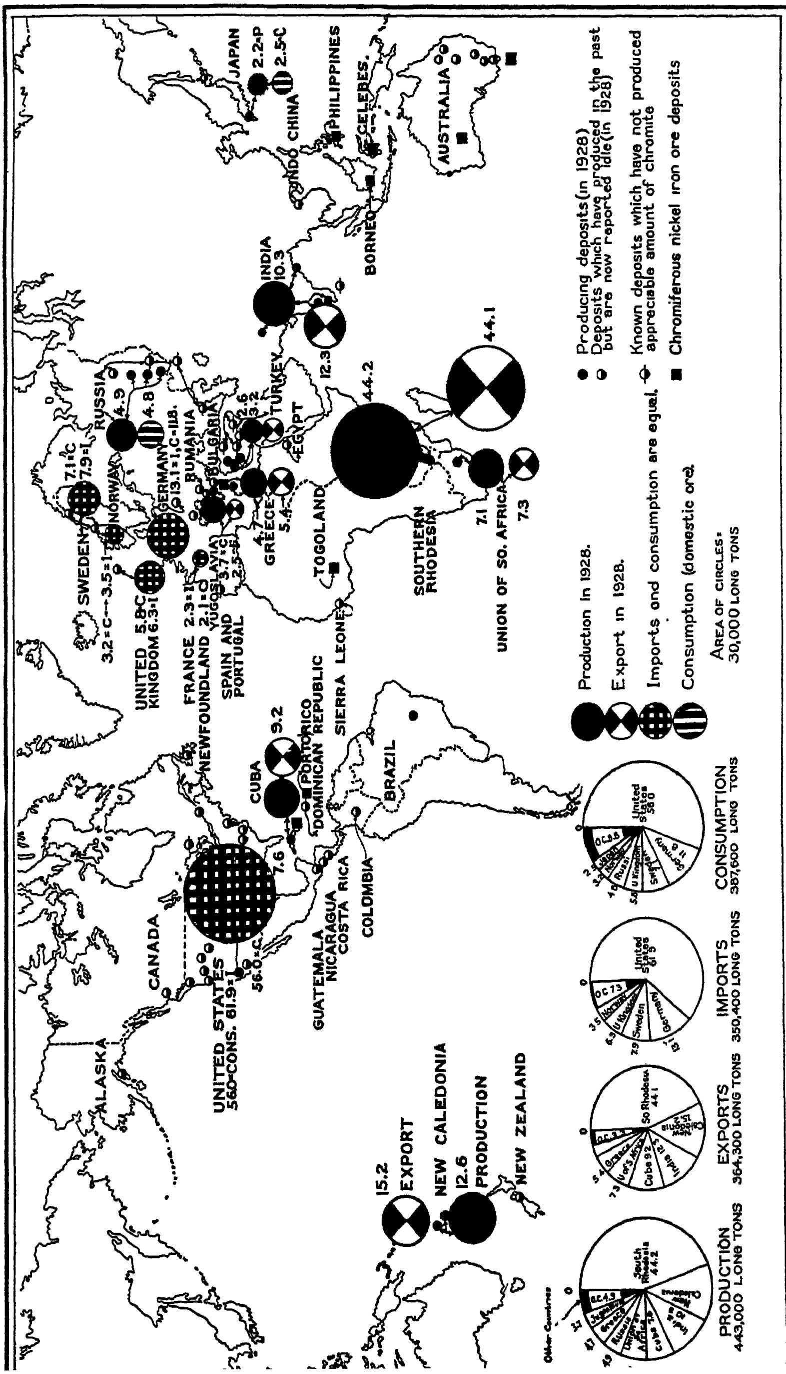

FIG. 3.—LOCATION OF WORLD CHROMITE DEPOSITS AND RELATIVE PRODUCTION, EXPORTS, IMPORTS AND CONSUMPTION OF THE PRINCIPAL PRODUCING AND CONSUMING COUNTRIES DURING 1928.

TABLE 3.—*Chromite Produced, Exported, Imported and Consumed by the Principal Countries in 1928*[a]

Country	Production, Long Tons	Per Cent. of Total	Exports, Long Tons	Per Cent. of Total	Imports, Long Tons	Per Cent. of Total	Apparent Consumption, Long Tons	Per Cent. of Total
Southern Rhodesia	195,900	44.2	162,300	44.1				
New Caledonia	56,000	12.6	56,000	15.2				
India	45,500	10.3	45,300	12.3			200	0.1
Cuba	33,700	7.6	33,700	9.2				
Union of South Africa	31,300	7.1	23,400	7.3				
Russia	21,600	4.9	3,000	0.8			18,600	4.8
Greece	20,600	4.7	19,800	5.4			800	0.2
Yugoslavia	16,400	3.7	9,100	2.5			7,300	1.9
Turkey	11,700	2.6	11,700	3.2				
Australia	700		[b]	[b]				
Japan	9,653	2.2			[c]	[c]	9,800	2.5
United States	660	0.1			216,600	61.9	217,300	56.0
Brazil	20		[b]	[b]				
Germany					45,800	13.1	45,800	11.8
Sweden					27,600	7.9	27,600	7.1
United Kingdom					22,400	6.3	22,400	5.8
Norway					12,400	3.5	12,400	3.2
France					8,000[d]	2.3	8,000[d]	2.1
Czechoslovakia					2,400	0.7	2,400	0.6
Other Countries	[e]	[e]	[e]	[e]	15,200[d]	4.3	15,000[d]	3.9
	443,000	100.0	364,300	100.0	350,400	100.0	387,600	100.0

[a] Producers and consumers stocks not taken into account.
[b] A few hundred tons of exports from Brazil and Australia unaccounted for.
[c] A few hundred tons imported into Japan included in the estimate for Other Countries.
[d] Estimated.
[e] Included in total.

WORLD RESOURCES

NORTH AMERICA

United States

United States production of chromite from 1827 to 1929, inclusive, was 478,000 long tons, or 6.7 per cent. of the world total. Although chromite was discovered in 1810, or 1811, by W. H. Hayden,[1] chromite mining in the United States began in 1827. During the period from 1827 to 1860, inclusive, this country led the world in production. It has been estimated that about 250,000 tons of ore[2] averaging about 40

[1] Baltimore Med. Phil, Lyceum (July 12, 1911) 1, 255–271.
[2] J. W. Furness: Chromite. U. S. Bur. Mines *Min. Resources of U. S.*, 1925 (1928) Pt. I.

to 63 per cent. Cr_2O_3, was shipped prior to 1880, practically all of which came from the Maryland-Pennsylvania field. In 1827 the Tyson Mining Co., through Isaac Tyson, began operations at the Reed mine, in the Soldier's Delight district, near Jarrettsville, Md. This ore was first manufactured into yellow and green paints and used in Baltimore for decorating carriages and furniture. Later a considerable amount was exported to Great Britain and other European countries for the manufacture of pigments and chemicals, but when the larger and more lucrative Turkish deposits were developed, the foreign market for Maryland ores was lost and chromite mining in the United States declined. From 1870 to 1880 the yearly production decreased, but recuperated slightly with the discovery and development of chromite in California around 1880, culminating in a production of 3680 tons in 1894. From 1900 to 1914, the annual average output was about 265 long tons, though imports steadily increased. The scarcity of shipping facilities during the World War caused the United States to turn to the deposits of Canada, Cuba, Brazil, Central America, and other countries of the Western Hemisphere for its supply of chromite. As these countries proved unable to supply requirements, an urgent demand arose for increased production from domestic deposits. The result was a production of 185,065 long tons of chromite in the United States from 1914 to 1919, inclusive, a peak production of 83,753 long tons being made in 1918. Most of this chromite was obtained from California and Oregon. Termination of the World War left United States producers with a large stock of high-cost chromite for which there was little demand.

In 1918 there were 450 shippers in the United States, distributed as follows: California, 374; Oregon, 60; North Carolina, 5; Montana, 3; Pennsylvania-Maryland, 3; Alaska, 3; Georgia, 1; Wyoming, 1; Washington, 1. In 1919 and 1920 some chromite from stocks on hand at the end of 1918 was shipped from domestic mines.

As production during the war had been requested by the government, a considerable sum was disbursed by a War Minerals Relief Commission in compensation for loss incurred by chromite producers. Since 1920, however, the yearly output of the United States has averaged about 230 tons and has never exceeded 660 tons.

Following are brief descriptions of the more important chromite areas of the United States, which are shown on Fig. 4.

Alaska.—The deposits of Alaska are found chiefly on the Kinai Peninsula. Reserves include about 50 deposits reported to contain 225,000 to 230,000 tons or more of ore or concentrates averaging over 40 per cent. Cr_2O_3.

California.—California has produced over 289,000 tons of chromite ore or concentrates, and latest reports indicate reserves of about 210,000 to 230,000 tons of ore or concentrates, averaging between 30 and 50

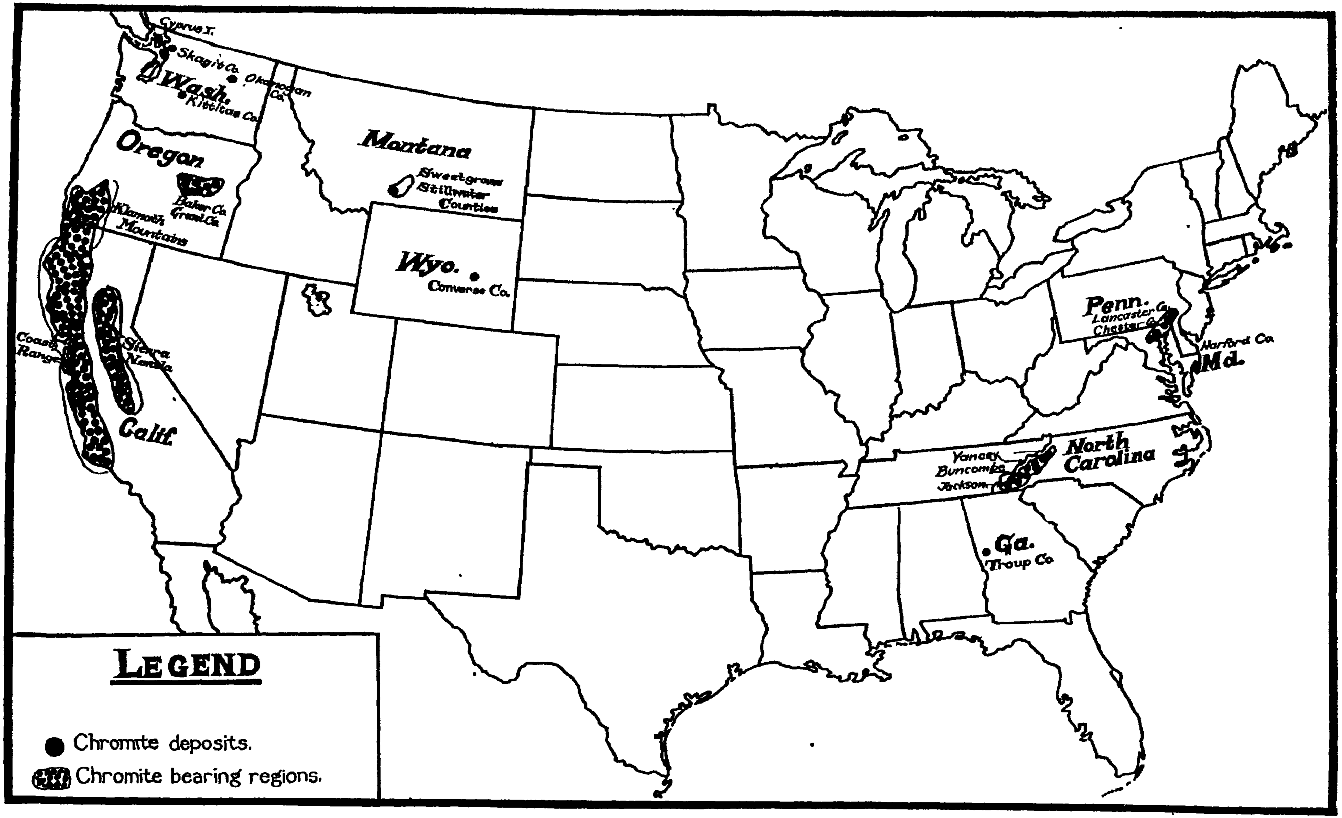

FIG. 4.—CHROMITE DEPOSITS OF THE UNITED STATES.

per cent. Cr_2O_3. There are over 860 reported deposits in the Sierra Nevada, Coast Ranges and Klamath Mountains. The ore usually occurs as lenses or disseminations in serpentine and serpentinized or schistose rocks.

Georgia.—The deposits at Louise, Troup County, Ga., have attracted some attention of late, but the reserves and grade of the ore are yet to be determined.

Montana.—A large chromiferous serpentine area lying in Sweet Grass and Stillwater counties is reported to contain a large tonnage of concentrating ore which may prove available for treatment. Most of the area has not been developed, but the west and east ends were under development during 1929. The serpentine belt is ½ to ¾ mile wide and 30 miles long. The reserve of this area has been reported[3] at about 200,000 tons, containing less than 40 per cent. Cr_2O_3, whereas the total Montana reserve in five deposits has been estimated[4] at about 500,000 tons of all classes of ore.

North Carolina.—North Carolina has produced about 400 tons of ore averaging around 45 per cent. Cr_2O_3 and reserves in five or six deposits in the western part of the state are reported at more than 1000 tons. The ore occurs as bunches or disseminations in dunite or serpentinized dunite.

Oregon.—There are two principal chromiferous areas in Oregon, one in the Klamath Mountains and the other in the Blue Mountains, as indicated on Fig. 4. Production from 1916 to 1925 was about 36,500 tons from 107 deposits, which are reported to have reserves amounting to more than 60,000 tons, averaging 40 per cent. Cr_2O_3.

Washington.—Deposits of chromite in Washington are located on Cyprus Island, Skagit County; near Mount Hawkins, Kettitas County, and in Akanogan County. Reserves are estimated at about 2000 tons of concentrates running 40 per cent. Cr_2O_3.

Wyoming.—Wyoming has one known deposit in Converse County which has produced over 1000 tons of ore averaging about 40 per cent. Cr_2O_3. The reserves are estimated at 1000 tons of concentrates running about 38 per cent. Cr_2O_3.

Canada

Nearly all of the chromite deposits of Canada are found in serpentines of Laurentian and Cambrian ages, distributed throughout Quebec and Ontario. Though there are numerous chromiferous areas, few deposits have proved of commercial importance. Most of the production has come from the Colraine, Thetford and Black Lake districts, in Quebec. Other deposits of chromite are located on the north slope of Taylor Basin, Bridge River district, British Columbia; in the Grand Forkes mining

[3] J. S. Diller: Chromite. U. S. Geol. Survey *Min. Resources*, 1918 (1921) Pt. I, 672.

[4] J. W. Furness: *Op. cit.*, 140.

district; on Scottie Creek, near Clinton; on Olivine Mountain, Tulameen district. Chromiferous iron ores, which carry 0.73 to 6.32 per cent. Cr_2O_3, are found in the Lake Webster region, British Columbia. Recently a discovery[5] was made south of Collin's station, on the Canadian National Railway and west of Armstrong, Ont. The ore containing 28 to 40 per cent. Cr_2O_3 is said to occur in a dike 700 ft. wide and about 1½ mile long.

The reserves of chromite in Canada have been estimated at several hundred thousands tons of low-grade ore and concentrates averaging about 40 per cent. Cr_2O_3.

Chromite was first mined in Canada in 1886 but not until 1894 was there a substantial output. From 1895 to 1901 the average annual output was about 2000 tons. This was increased to over 8000 tons in 1906, but by 1912 production had ceased completely. The shortage of chromite in the United States during the World War resulted in a revival of chromite mining in Canada, and from 1915 to 1920 the production ranged from 8000 to 33,000 tons, the record yearly production being 32,791 tons in 1917. The industry declined again, and since 1924 there has been no production. (The total production of Canada since 1880 has been over 174,000 tons, about 2.5 per cent. of the world total since 1827.)

Costa Rica[6]

Shipments of high-grade ore to the United States were made by the United Fruit Co. during 1918, but there has been no production since the World War. It is reported that the deposits are large and further development may yield more ore.

Cuba

Production of chromite in Cuba began in 1916, and the total output from 1916 to 1929, inclusive, as indicated by imports from Cuba into the United States, was about 213,000 tons, or about 3.0 per cent. of the total world output from 1827 to 1929, inclusive.

The deposits of Cuba described by Burchard[7] in 1919 are in Camaguey, Oriente and Matanzas provinces. The most important mines lie in the Nuevitas district, in Camaguey, although considerable production has come from several mines around Nipe Bay (Oriente). In addition, there are extensive deposits of chromiferous nickel-iron ores in Oriente and Camaguey provinces. The reserves of chromite have been estimated at around 100,000 tons, and there is reported to be 3,000,000,000 tons of chromiferous-nickel iron ore in the Mayari and Moa Bay fields. The

[5] Skilling's *Mining Review* (Oct. 13, 1928) 8.

[6] Chrome in the Caribbean Countries. *Iron Age* (1920) **105**, 1509.

[7] E. F. Burchard: Chrome-ore Deposits in Cuba. *Trans.* A. I. M. E. (1920) **63**, 150.

chromite in Cuba is low grade, averaging about 33 to 43 per cent. Cr_2O_3, 10 to 12 per cent. Fe, 1 to 6 per cent. SiO_2, 20 to 32 per cent. Al_2O_3, and some moisture. The magnesia content may reach 14 to 20 per cent. in many ores. The chromiferous nickel-iron ore averages around 42 to 47 per cent. Fe, 2 to 7 per cent. SiO_2, 6 to 12 per cent. Al_2O_3, 1 to 2 per cent Cr, 12 to 14 per cent. moisture and up to 0.7 per cent. Ni. The sintered iron ore averages 55 to 56 per cent. Fe, 4.5 per cent. SiO_2, 12 to 14 per cent. Al_2O_3; 2 per cent. Cr, up to 1 per cent. Ni and low phosphorus and sulfur.

Newfoundland

Chromite deposits on the west coast of Bluff Head,[8] Port au Port Bay, 30 miles from Sandy Point, were worked from 1895 to 1900, inclusive. The total production of about 5500 long tons was shipped to Philadelphia, where it was made into chromite brick. This ore was low grade and required concentration. Chromite has been found at Benoit Brook and near the headwaters of Bay d'Est and on the Gander River.

Nicaragua

Small quantities of chromite were reported to have been shipped from Nicaragua during the war. No definite information about the occurrence of the ore has been obtained.

Porto Rico

There is a deposit of limonitic iron ore associated with serpentine at Mayagerez, on the west coast. Reserves of 430,000,000 tons of material similar to the Mayari ore are reported. The ore contains 57.7 per cent. Fe_2O_3, 20 per cent. Al_2O_3, 2.4 per cent. SiO_2, 1 per cent. NiO and 1.57 per cent. Cr_2O_3. This ore can be nodulized, as can the ores of the Mayari field.

South America

Brazil

Chromite mining began in Brazil during the World War, but ceased shortly afterward. The Santa Luzia mine[9] in the vicinity of Queimadus and Bom Fim was the principal source of production, and over 26,000 tons of ore averaging 44 to 46 per cent. Cr_2O_3 was mined and exported to the United States. (Other deposits have been located at Campo,

[8] W. G. Rumbold: Chromium Ore. *Monograph* Imp. Inst., London (1921) 29–30.

[9] The Mineral Deposits of South America, Ed. 1. New York, 1919, McGraw-Hill Book Co.

Formoso and Saunde.)[10] The shipments of chromite ore in Brazil from 1918 to 1929, inclusive, amounted to 30,200 long tons, of which 60 per cent. was shipped in 1918.

Colombia

Large deposits of chromite have been reported at Antioquia,[11] where the ore was used as a building stone. In 1866 and 1867 a French company at Medellin smelted the iron and chromite ores of this region, producing a chromium iron of the following composition: Cr, 1.95 per cent.; C, 4.40; Si, 0.75; P, 0.07; S, tr; Va, tr; Mn, 0.84; Fe, 92.50; total, 100.51. Other analyses indicated 2.80 and 3.80 per cent. chromium.

Europe

Austria-Hungary

Prior to the partition of Austria-Hungary in 1919 some chromite was produced from deposits in Bosnia and Banat, the former being ceded to Yugoslavia and the latter to Rumania. During the World War the deposits near Sarajevo in Bosnia and Orsova in Banat produced considerable chromite. In present Austria, deposits, which were worked during the nineteenth century, occur in a belt of serpentinized peridotite that runs along the Mur Valley from Kraubath northward to Loeben, in the province of Styria.

Bulgaria[12]

Chromite deposits in serpentine have been found at Sotir and Ferdinandovo in western Bulgaria, but as yet they have not been developed. The ore is said to be of a fairly good grade. (See Fig. 5.)

Greece

The principal deposits in Greece are in Thessaly and on the Khalkidike Peninsula. Numerous other smaller deposits occur on the south side of Lake Daoukli, at Nezero; in the Neochori district, in the Othyris Mountains; at Voivoda; at Katerini in Macedonia (20 miles west of Salonika); at Euboea, at Boeotia and on the Island of Skyros (Fig. 5).

The total production from Greece has amounted to about 336,000 long tons, about 3.0 per cent. of the world total since 1827. The production, most of which is exported to the United States, amounted to 20,622 tons in 1928, compared to 14,813 tons in 1927.

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[10] H. Brett: Mineral Resources of the State of Bahia. Consular Rept. 103182 (July 26, 1923).

[11] M. Boussingault: Note on the Production, Constitution, and Properties of Chrome Steel. *Annales de Chemie et de Physique* [5] (1878) **15**, 91. (Reprinted and translated, *Jnl.* Iron and Steel Inst. (1886) No. 2, 813–814.

[12] W. Bruno: Beitrag zur Kentniss der Erzlagerstatten Bosniens. Sarajevo, 1887.

Germany

Germany's resources of chromite under present market conditions are negligible. A large deposit of low-grade ore occurs in Upper Silesia, on the southern slope of Mount Zobten[13] between Schweidnitz and Jordansmuehl. The better grades of this ore run 34 to 42 per cent. Cr_2O_3, 19 to 22 per cent. Fe_2O_3, 19 to 22 per cent. Al_2O_3, 16 to 18 per cent. MgO and 3 to 5 per cent. SiO_2. As the ore is unsuited for refractories the deposit has not been worked to any extent.[14]

Other deposits carrying 19 to 26 per cent. Cr_2O_3 were discovered in 1887 on the northern flank of Haarteberg, near Grouchau. A small amount of 50 per cent. ore was mined at times.

Large imports of chromite ore are thus necessary to supply a domestic consumption, which requires about 12 per cent. of the world total production. Imports were 42,000 tons in 1929, 46,600 metric tons in 1928, and 37,200 tons in 1927.

Italy

A chromite deposit[15] has been reported near Ziona, in the Upper Vara Valley, but no production has been made thus far. Italy imports its chromium chiefly in the form of bichromate and ferrochromium, although some chromite appears to have been exported to Italy in 1928.

Norway

Chromite has been mined in Norway since 1820, but little ore is now produced, and chromite must be imported to supply the bichromate and ferrochromium manufacturing industries. In 1928, Norway imported 12,640 long tons of ore for consumption. The total recorded production of chromite in Norway from 1830 to 1929, inclusive, was about 18,000 long tons.

Vogt[16] has described the Norwegian deposits in detail and states that the chromite of the Nordland (on the Hestmando and Roro Islands) is usually associated with peridotite, of the saxonite variety, and that in the Trondhjem district (at Trondhjem) it occurs in serpentine derived from peridotite.

Rumania

In southern Banat[17] near Orsova, 2 to 10 km. from the Danube River, Dubova and Pavisevita are chromite deposits for which large reserves are

[13] Mineral Industry, 1893, **2**, 156.

[14] *Stahl u. Eisen* (1890) **10**, 1085; (1891) **11**, 643.

[15] Soc. Geol. Ital. *Bull.* (Rome) (1924) **43**, No. 2, 183–188.

[16] J. H. L. Vogt: Zur Classification der Erzvorkommen. *Ztsch. f. prakt. Geol.* (1894) **2**, 384–393.

[17] *L'Industrie Chimique*, Paris, 16th year (Sept., 1929) No. 188, 1 (Appendix).

claimed. The Germans mined about 40,000 tons during the war, and Rumanian engineers have estimated reserves at several million tons of probable and possible ore. (See Fig. 5.)

Portugal[18]

In the neighborhood of Braganca, chrome deposits have been found which are reported to contain sufficient reserves to warrant exploitation, although some treatment may be required to bring part of the ore up to the grade of the Rhodesian or other high-grade ore.

Russia

Comprehensive data on the chromite resources of Russia are not available, but owing to the large areas of basic rocks, in which chromite is often found, it is probable that the resources of Russia will prove to be large. The Soviet government has been very active in prospecting for and developing new deposits.

Deposits of chromite are known in the Ural, Northern Caucasus, Transcaucasia and the Hinter Baikal regions. The most important of the Ural deposits include the Saranovsky and Gologorsky, and others in the Zlatoustovsky, Ubalinsky, Redinsky, Miassky and Troitsky districts, south of Sverdlovsk on the east side of the Ural Mountains. The grade of Russian ores usually is relatively lower than that of Indian, Rhodesian, and New Caledonian ores, ranging from 20 to 41 per cent. in the Gologorsky district and from 40 to 46 in the Saranovsky district. Only the Ural deposits have been commercially developed, and their reserves are estimated by the Soviets[19] to exceed 6,000,000 tons of various classes of ore. The chromite is found in lenses, stringers and disseminations in serpentines and soapstones and the chromiferous area extends along the east side of the Ural Mountains from north of Sverdlovsk southward into Orenburg.

The production in 1928–1929 was 28,000 long tons, compared with 22,000 tons in 1927–1928 and 19,000 tons in 1926–1927. The production in 1913 amounted to about 25,000 tons, practically all of which was obtained from the Ural region. Russia exports only 2000 to 4000 tons, the bulk of the supply entering into the production of bichromates or directly into the manufacture of steels. The total output of chromite from Russia is not known, but has been roughly estimated at about 650,000 tons, or about 9.2 per cent. of the world total from 1827 to 1929, inclusive.

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[18] The Iron and Chrome Ores of Northeast Portugal. *Min. Jnl.* (London) (1929) **164**, 156.

[19] Ann. Rev. of the Min. Res. of U. S. S. R. (1926–1927) 1054–1060.

Turkey

The chromite deposits of Turkey were discovered in 1848 and Turkey was the leading source of chromite ore from 1860 until the New Caledonian deposits came into large production in 1896. The total output to the end of 1929 was about 832,000 long tons of relatively high-grade ore (or about 11.7 per cent. of the world total since 1827). The principal deposits are shown on Fig. 5.

The most important area includes the mining towns of Brusa, Eskisehir, Kosfindik, Kutahia, Tavshanly and Balikeser, and is located about 70 km. south of Stambul. Other important deposits have been reported near Smyrna, Denizli, Makri, Budrum (Bay of Kos), Marmaras (Gulf of Makri), Adalia, Mersina, Adana, Tash-Tepe, west of Angora, Kastamuni and near Diabekir. For a further recent description of the Turkish deposits, reference may be made to a paper by Von Englemann.[20]

Table 4 shows the production of chromite in Turkey from 1881 to 1925 by five-year periods, and for 1926, 1927, 1928 and 1929 in long tons.

TABLE 4.—*Production of Chromite in Turkey, from* 1881 *to* 1929

Period	Production, Long Tons	Annual Average Production, Long Tons	Period	Production, Long Tons	Annual Average Production, Long Tons
1881–1885	47,542	11,886	1916–1920	71,008	14,201
1886–1890	97,722	19,544	1921–1925	23,075	5,769
1891–1895	117,812	23,562	1926	6,565	6,565
1896–1900	56,075	11,215	1927	18,029	18,029
1901–1905	53,148	10,630	1928	11,662	11,662
1906–1910	109,248	21,850	1929	14,840	14,840
1911–1915	105,774	21,155			

United Kingdom

The United Kingdom is an important consumer of chromite, but most of its needs are supplied by imports. A small and intermittent domestic supply has been obtained from Scotland, where chromite occurs at several places on the mainland and on the Shetland and Hebrides Islands. The Island of Unst, of the Shetland group, is the only locality that has produced substantial quantities. Here the chromite occurs in lenses and disseminations in serpentines. Exploitation began as early as 1820, and ore running 44 to 48 per cent. has been extracted, though the larger part of the material requires concentration. The concentrates

[20] H. Von Englemann: Turkey Extending Railroads to Develop Chrome Deposits. *Eng. & Min. Jnl.* (1929) **127**, 1037–1038.

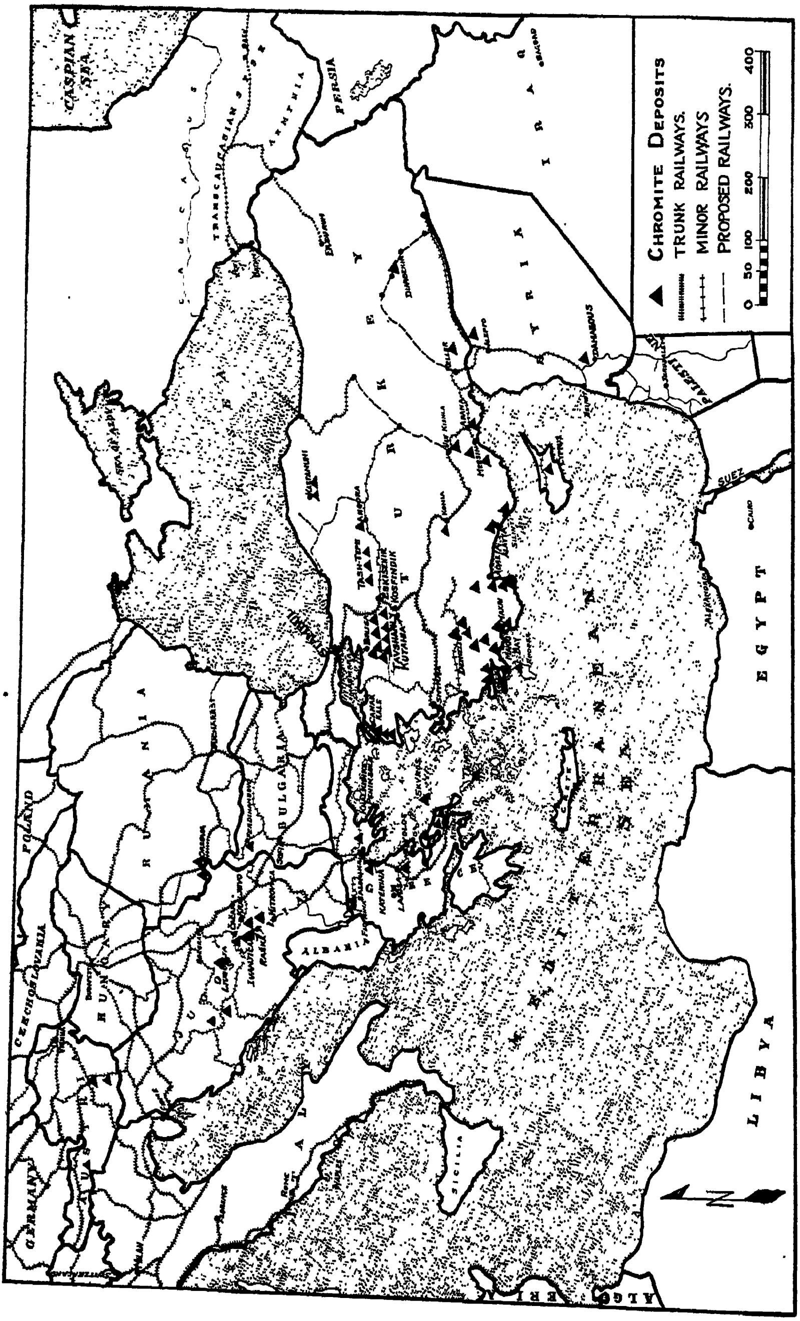

FIG. 5.—CHROMITE DEPOSITS OF ASIA MINOR AND THE BALKAN STATES (TURKEY IN ASIA, GREECE, YUGOSLAVIA, BULGARIA, RUMANIA, AUSTRIA, CYPRUS, ETC.).

in recent years have averaged 32 to 34 per cent. and have been used for refractories. The deposits have been described by Hitchen.[21]

Yugoslavia

The provinces of Bosnia, Serbia and Banat in Yugoslavia produced an annual average of about 19,320 tons during the five-year period from 1925 to 1929, inclusive. During 1929 the output amounted to 43,022 metric tons as compared with 16,678 tons in 1928 and 8757 tons in 1927. Diller[22] has described the deposits of Bosnia and Serbia for 1918. (See Fig. 5.) The principal deposits are in eastern Bosnia and northcentral Serbia, and extend from north of Vares (Bosnia) into the Kopaonik Plataeu south of Cačač.

Asia

India

The principal deposits of chromite in India are found in the Quetta-Pishin and Zhob districts in Baluchistan; the Hassan and Mysore districts in Mysore State; and the Singhbhum district in the states of Bihar and Orissa. The deposits in Baluchistan occur in veins and irregular segregations in the serpentine, which accompanies the great basic intrusions of Upper Cretaceous age. Much of the ore averages in excess of 50 per cent. Cr_2O_3. In Mysore the ore is found in ultra-basic dunite dikes, cutting gneiss, which have been altered in places to serpentine, and it generally occurs in veins ranging from 9 to 12 in. in width. The ore usually averages up to 52 per cent. The Hassan ores lie in a talc-serpentine belt about 20 miles long, and the ore is mainly in disseminations of low tenor. The Singhbhum deposits often contain up to 53 per cent. Cr_2O_3 and occur in a series of ultra-basic igneous rocks. Other promising deposits have been reported in Bombay Presidency and on the Island of Ceylon. Indian reserves in 1925 were estimated to contain about 800,000 long tons of ore, averaging 50 per cent. Cr_2O_3.

During the last four years, 1926 to 1929, inclusive, production has come from the Zhob, Hassan, Singhbhum and Mysore deposits of which Zhob and Hassan are the most important. The production from 1903 to 1929 has totaled about 616,000 long tons, or 8.7 per cent. of the world total since 1827. Production in 1929 amounted to 49,566 long tons compared with 45,456 in 1928.

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[21] C. S. Hitchen: Unst and its Chromite Deposits. *Min. Mag.*, London (1929) 40, 18.

[22] Chromite. *Min. Res. of U. S.*, 1918 (1920) Pt. I, 685–687.

Indo-China

Extensive deposits[23] of low-grade ore occur 25 km. southwest of Thank-Hoa (Annam) in the valleys lying between the Mountains of Nui-Na-Son and Nui-Nua. A survey of these deposits, completed in 1924, showed alluvial material containing 11.6 to 22 per cent. Cr_2O_3. The deposits are estimated to contain 1,700,000 tons of ore which can easily be mined and milled. The only recorded production was 20 tons during 1924 and 11 tons in 1929.

East Indies

Netherland East Indies, Borneo.—Chromiferous iron ores are located on the Island of Seboekoe and at Soengei Doewa in the southeastern tip of Borneo on the slopes of Mount Koesambi. The ore is similar in character to the Mayari ores of Cuba. The reserves at Seboekoe are reported to contain 300,000,000 tons, and at Soengei Doewa, 100,000,000 tons. The Seboekoe ores contain from 50 to 53 per cent. iron, 2 to 2.3 per cent. chromium, 0.39 to 0.45 per cent. nickel, and are low in silica.

Philippine Islands

Mindanao Island.—Lateritic iron ores similar to the Mayari ores of Cuba occur on the coast of Surigao Province on the Island of Mindanao. This orebody covers an area of about 40 square miles, stretching along the coast for about 10 miles. It varies in thickness from 1 to 60 ft. The ore is a reddish to yellowish brown hydrated oxide and carries about 47.40 per cent. iron when dried. Silica, phosphorus and sulfur are low; alumina runs from 10 to 12 per cent. and the total water content is about 20 per cent. Reserves are estimated at 800,000,000 tons by J. F. Kemp and at 500,000,000 tons by the American Chamber of Commerce of the Philippine Islands. Favorable localities in the lateritic iron ore region may have high-grade chromite deposits, and it is understood that prospecting is now in progress.

AFRICA

Egypt

A small deposit of chromite in the eastern desert of Gebel Hagar Dungash was reported by Stewart[24] in 1904. Analyses indicate that the material averages around 30 per cent. Cr_2O_3.

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[23] Position and Prospects of Chromium. *South African Min. and Eng. Jnl.* (1928) **39**, 122.

[24] Report on Mineral Resources of Egypt, 36. Govt. Press, Cairo, 1922.

French Guinea[25]

Chromite ore in peridotite has been found in an area south of Kaloulima.

Sierra Leone[26]

A number of lenticular deposits have been disclosed near the village of Gerihum, 6 miles north of Hangha and 188 miles by rail from Freetown. The exposures are only 3 miles west of the motor road from Hangha to Lago. The ore is said to contain from 39 to 48 per cent. and average 45 per cent. Cr_2O_3. Another deposit was found 2½ miles south of Senduma, 15 miles by road from Blama and 169 miles by railroad to Freetown. This ore is said to contain from 31.4 to 43.8 per cent. and to average 36.4 per cent. Cr_2O_3. The region has not been adequately prospected and it is not improbable that further exploration will disclose other valuable deposits.

Southern Rhodesia

Southern Rhodesia is the world's largest producer of chromite, supplying nearly one-half of the world output. Production in 1929 amounted to 261,710 long tons, compared with 195,916 in 1928. Of the 1928 production, the Gwelo district produced 79 per cent.; the Lomagundi district, 16 per cent.; the Victoria district, 5 per cent.; and the Hartley district, a smaller proportion. The total production of Southern Rhodesia from 1906 to 1929, inclusive, was 1,912,000 long tons. The production comes largely from deposits of high-grade ore which occur along the Great Dyke, a formation of basic and ultra-basic rocks averaging 4 miles wide, and traversing the central part of the country in a north-northeast and south-southwest direction for 330 miles, as shown on Fig. 6.

The Victoria district is the southernmost of the chrome-producing areas. The principal deposits are at Mashaba, several miles east of the Great Dyke. Lack of cheap transportation has handicapped the development of this district, but in 1929 the railway extension from Victoria was completed.

The principal deposit of the Gwelo district, which lies northwest of the Victoria district, is at Selukwe. This deposit has been producing steadily since 1906 and has supplied a large part of the Rhodesian output. The ore occurs in lenses in talc schist, chlorite schist and serpentines. Most of the lenses being exploited range from 150 to 450 ft. in length.

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[25] W. J. Yearly: Consular Rept. 672 (Sept. 14, 1920) Dakar, Senegal.

[26] N. R. Junner: Geology and Mineral Resources of Sierra Leone. *Min. Mag.* (London) (1930) **42**, 73–82.

The following is said to be a typical analysis of Selukwe ore:[27] Cr_2O_3, 51.10 per cent.; FeO, 11.40; Fe_2O_3, 1.40; MnO, 0.50; Al_2O_3, 15.20; MgO, 12.70; CaO, 0.90; P_2O_5, 0.50; S, trace; SiO_2, 4.80; H_2O, etc., 1.97.

Another important producer of the Gwelo district is the Lalapanzi area, about 30 miles northeast of Selukwe. The chromite here occurs in three continuous seams paralleling the Great Dyke. The seams dip at an

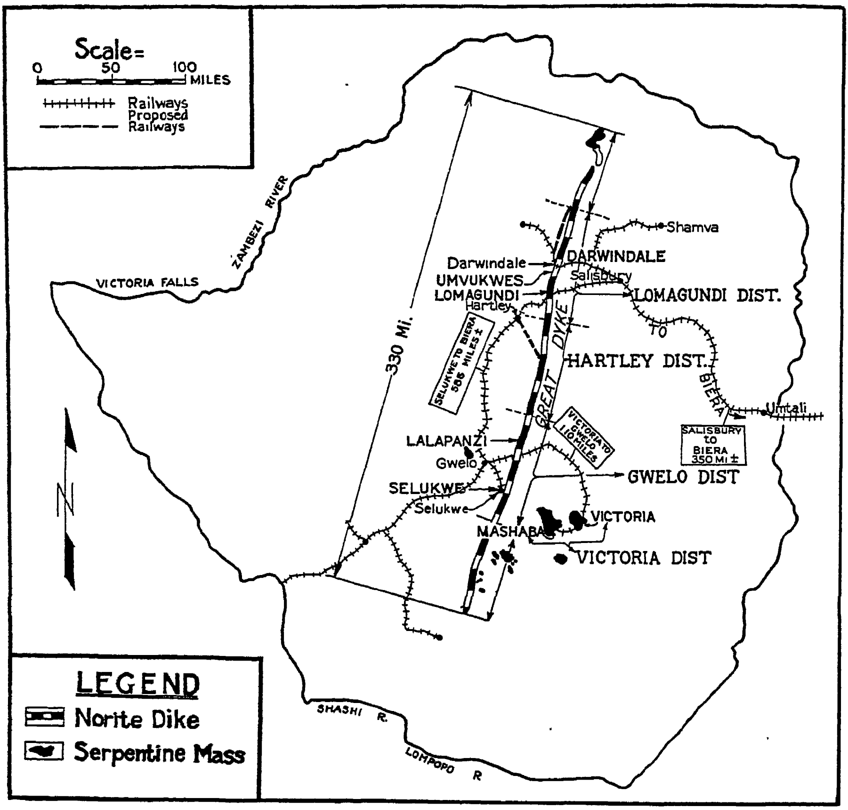

FIG. 6.—CHROMITE DEPOSITS OF SOUTHERN RHODESIA.

angle of 25° and are from 2 to 9 in. wide. The grade of the ore ranges from 48 to over 50 per cent. Cr_2O_3. The Gwelo district has adequate rail transportation to the port of Biera, Portuguese East Africa.

The Hartley district lies about 80 miles north of Lalapanzi. In 1929 the Bee Chrome Co. controlled 220 blocks of claims in this area, and is reported to have uncovered three seams which appeared to be continuations of three of the Lalapanzi and Umvukwes on the south and north,

[27] J. W. Furness: Chromite. . U. S. Bur. Mines. *Min. Resources* of the U. S., 1927 (1930) Pt. 1, 319.

respectively. The seams dip 10° on both sides of the dike, forming a trough. Two of these seams average 8½ in. in width over a large area. The reserves as developed over a length of 43 miles and an average inclined depth of 100 ft. were estimated at about 3,000,000 tons averaging 49.3 per cent. Cr_2O_3. The ore is transported 43½ miles to Hartley station by trucks and wagons, but a narrow-gage railroad to better transportation facilities has been proposed. It was estimated that the cost of delivering ore, c.i.f. European and United States ports, would range between $18 and $19.[28]

The Lomagundi district, which lies north of the Hartley area, is the second largest producer of chromite in Rhodesia. Several chromite

TABLE 5.—*Production of Chromite in Southern Rhodesia from* 1906 *to* 1929, *Inclusive*

Period	Quantity, Long Tons	Period	Quantity, Long Tons
1906	3,256	1918	27,934
1907	10,415	1919	31,502
1908	11,927	1920	53,812
1909	22,875	1921	44,811
1910	39,288	1922	83,460
1911	46,753	1923	86,317
1912	61,839	1924	154,218
1913	56,593	1925	121,274
1914	43,042	1926	161,781
1915	54,089	1927	194,660
1916	79,349	1928	195,918
1917	65,145	1929	261,711

seams have been found in the Dyke, which extend almost continuously from Darwindale to the vicinity of the farm Gurungwe, a distance of approximately 70 miles. The continuity of the deposit apparently is broken near Gurungwe, but ore occurs again further north and is said to be plentiful and of high quality. Owing to lack of rail transportation, production in this district has been confined largely to the area immediately north of Darwindale, where the railroad crosses the Great Dyke. Recently deposits at Umvukwes, several miles north of Darwindale, have been developed and appear to contain large quantities of high-grade chromite. Early in 1930 the construction of a branch railway from a point near Darwindale northward along the west side of the Dyke was reported. The completion of this railroad is expected to result in a greatly increased production in the Lomagundi district.

[28] Important Rhodesian Chrome Development. *South African Min. and Eng. Jnl.* (1929) **40,** 451–453.

Mr. Keep,[29] of the Southern Rhodesia Geological Survey, discussed the reserves and future of the Umvukwes deposits.

Lack of data makes it impossible to appraise numerically the reserves of chromite of Southern Rhodesia, but it can safely be stated that the deposits now developed along the Great Dyke are capable of producing several million tons of high-grade ore. Table 5 gives the production of chromite in Southern Rhodesia from 1906 to 1929, inclusive.

Union of South Africa

Transvaal.—Large deposits of chromite have been developed recently along the margin of the Bushveld Complex at Rustenburg and Lydenburg. Accurate data on the reserves in this area are not available, and estimates which have appeared in the literature vary greatly. However, it is probable that there are several million tons of ore in these deposits. The ores for which assays are available are comparatively low grade, averaging about 43 per cent. Cr_2O_3. Owing to the lower transportation and mining cost, the ore can be marketed at a cost per unit of chromium considerably below the Rhodesian ores. It has not, however, been demonstrated that consumers will remodel metallurgical plants, originally designed to handle 50 per cent. ore, in order to take advantage of the cheaper Transvaal chromite. At present the ore finds a market in the chemical and refractory industries, and some of the higher grade ore has been used in the metallurgical industry. Schneiderhohn has described the Bushveld deposits.[30]

The production from 1921 to 1929, inclusive, in long tons was about 138,000, of which 96 per cent. was obtained during the period from 1925 to 1929, inclusive, and 46 per cent. in 1929.

Natal.—Chromite has been found in the serpentines on the farm at Tugela Rand, near Krantz Kop, Natal. Samples from this deposit contained 25 to 28 per cent. Cr_2O_3, too low grade to be mined profitably at present.

Togoland.—In 1908, Koert[31] reported a deposit of chromite situated near the railway line from Lome to Atakpame. The ore is reported to run 36.7 to 41.7 per cent. Cr_2O_3.

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[29] F. E. Keep: Interim Report on the Geology of the Chromite Deposits of the Umvukwe Range, Lomagundi District. Southern Rhodesia Geol. Survey *Short Rept.* 23 (1928); reference in *Min. Mag.* (London) (1929) 181.

[30] H. Schneiderhohn: Der XV. Internationale Geologenkongress in Südofrika. Der Eruptivkörper des Bushvelds und Seine Chrom-, Nickel-, Platin- und Zinnlagerstätten. *Metall Wirtschaft* (1930) **9**, 273. Also *Min. Jnl.*, London (1930) **169**, 345.

[31] Koert: Über ein Chromeisensteinvorkommen im Atakpamebezirk. *Geol. Zentralblatt* (1908) **11**, 707. Abs.

OCEANIA

Australia

Production of chromite in Australia has been relatively negligible. Mining began in 1882 and from 1894 to 1901 over 3000 tons was produced annually. Since then the annual output has seldom exceeded 1000 tons, and in 1929 about 130 tons was produced as compared to no production in 1928. The total output from 1880 to 1929, inclusive, exceeded 41,000 tons. Production has come largely from New South Wales and Queensland, whereas deposits also have been reported in Victoria, Western Australia and Tasmania. The ore from New South Wales comes from Bowling Alley Point, Clarence River, Bingora, Barraba, Port Macquarie (beach-sand) and the Tamworth districts in the northern area, and from the Vulcan, Quilters, Mount Mary, Kangaroo and Mount Miller mines in the southern or Gundagai-Tumut area. The deposits of Queensland are in a belt of serpentine extending from Keppel Bay to Marlborough. Reserves were estimated at about 500,000 tons in 1925.

New Zealand

New Zealand produced small quantities of chromite prior to 1903, chiefly from the Nelson mining district. Other deposits are in Westland and Otago.

New Caledonia

Chromite has been mined in New Caledonia[32] since about 1875 and the production up to the end of 1929 has totaled about 1,347,000 long tons, or about 19 per cent. of the world total since 1827. Exports in 1929 amounted to 58,212 long tons. There were five[33] operators during 1929, the most important of which was the Societa La Tiebaghi, which operates the Tiebaghi mine at Pagoumene. This mine produces ore carrying 48 to 52 per cent. Cr_2O_3. It is equipped with modern ore-handling devices, but operating costs are high owing to depth. Société Chimique du Chrome operated the Fantoche and Alpha mines in the Pagoumene district. The other operations were those of the Société Caledonia which has mines at Coulee and Plaine des Lacs, that of M. Talou with the Chagrin mine at Koumac, and that of M. Vernier who operated the Alice-Louise mine on the Bay of N'Go. The mines are controlled chiefly by British and American capital. The ore occurs in serpentinized peridotite, and in 1925 reserves were estimated[34] at more than 1,500,000 tons of 50 per cent. ore.

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[32] J. S. Diller: *Op. cit.*, 704–709.

[33] Chrome Mining in New Caledonia. Commerce Rept. 25 (June 24, 1929) 781.

[34] J. W. Furness.: Reference of footnote 2, 145.

World Reserves and Resources of Chromite

World reserves of chromite are hardly susceptible to precise estimation. It seems apparent that Southern Rhodesia, Turkey, India, Union of South Africa, Russia, Canada and New Caledonia should be able to meet world requirements of chromite for several decades. Beyond that, uncertainties dependent upon technological development, price and other factors arise. To what extent, for example, may low-grade ores of various types be regarded as reserves? Of chromiferous iron ores of the Mayari type, possibly as much as 5 billion tons exist in Cuba, Porto Rico, Celebes, Borneo, Philippine Islands, Gold Coast, Greece and Australia; but such ores cannot at present be regarded as reserves, though the chromite they contain is likely to be utilized more and more extensively as exhaustion of relatively high-grade ore occurs. To ores of intermediate grade, special technologic problems often apply that depress their present economic status without precluding hope of its improvement. These and the very low-grade ores provide ultimate resources of chromite but cannot as yet be regarded as reserves suitable for inclusion with the high-grade ores now required by industry.

Political and Commercial Control of Deposits

The world reserves of chromite are chiefly controlled by British, German, American, Russian and French capital, in order of importance. Swedish capitalists are strongly interested in the Turkish deposits.

North America.—The deposits of the United States, Cuba, Porto Rico, Guatemala and Nicaragua are operated or controlled by American capital, while those of Canada and Newfoundland are controlled by both American and Canadian interests.

South America.—During the World War the Brazilian deposits were worked by an American concern, but their present status is uncertain. Little is known of the Colombian deposits.

Europe.—The deposits of chromite in Austria, Germany, Italy, Norway and the United Kingdom are owned by citizens of these countries, and those of Russia are controlled and operated by the Soviet Government. The companies, Société de Recherches et Travaux Minière Vaudos, at Vaudos, and the Société Union Minière at Larissa, control most of the Greek properties; the remainder are controlled or operated by American and Greek companies.

The Rumanian deposits were worked by the German Military Mission during the war, but the Rumanians apparently have assumed control since that time. In Yugoslavia two German companies, the German Meteor Group and the Ljubotan (67 per cent. German and 33 per cent. Yugoslav) operate some of the principal mines. Other operators include

the Allatini Brothers, Jaques Heil, Hoope & Co., the Hemicros Group and a newly organized French company.

The Japanese chromite deposits are controlled and operated by Japanese citizens, while the low-grade deposits of Indo-China apparently are conceded to a French company. The Indian deposits are largely controlled by British capital.

In Turkey, where reserves are reported to amount to several million tons, brisk competition has developed for concessions. The Roeschling Group (German) has acquired large holdings in the Dagh Ardy sector of the Brusa field. The eastern half of this field is jointly controlled by the Edmond Davis Group, a syndicate of Swedish Iron Works, and the German Krupp Works. A French group has become interested in the Eskisehir district, while mines farther north, in the Bolsular district, have been conceded to a Turkish company (recent reports indicate that the Roeschlings have acquired the holdings of the Turkish concern). The Makri deposits are operated by a French company, Société Minière de Fethie, while the Smyrna mines have been conceded to the Patterson Brothers (British). American capital is said to be negotiating for a concession to a certain area in the Brusa district, and two American concerns are reported to have concessions which are under development or exploitation.

Africa.—The large chromite reserves of Southern Rhodesia and the Union of South Africa are owned mainly by the Chrome Corporation, Ltd. (the Edmond Davis Group (British). Less important areas are being developed by the Bee Chrome Corporation, Ltd., Mann-Little & Co., and the Becker Trust Co. (all British). American interests are reported to own considerable stock in certain of these companies.

Oceania.—In New Caledonia the Tiebaghi mine at Pagoumene is operated by the Société La Tiebaghi (Edmond Davis Group (British)), and the Fantouche and Alpha mines in the same district are being exploited by the Société Chimique du Chrome (American). The Société Caledonia (nationality not determined) works a number of mines at Coulee and Plaine des Lacs. There are also a number of small French and native operators.

The deposits of Australia and New Zealand are owned by British or Australian capital.

DISCUSSION

(Albert O. Hayes presiding)

L. D. Huntoon, New York, N. Y. (written discussion).—In mentioning the chromite deposits of Bluff Head, Port au Port Bay, west coast of Newfoundland, and the operation in 1895, Mr. Smith apparently was unfamiliar with the work done by George W. Maynard and the "French Shore" situation at that time. In 1894 Mr. Obalski, mining engineer for the Province of Quebec, examined the deposits and submitted a

favorable report. In September, 1895, George W. Maynard examined the property and in the following year explored the property and submitted a favorable report. He reports that assays of different exposures range from 39 to 50 per cent chromic oxide; further, that 146 tons were shipped to Philadelphia, which assayed 49.90 chromic oxide, and that close cobbing will produce a 50 per cent product. Gravity concentration tests were made by Prof. R. H. Richards on ore assaying 47.94 per cent chromic oxide. The result was reported to have been concentrates assaying 55.30 per cent chromic oxide and a recovery of 88.5 per cent. Although reports by both Mr. Maynard on the deposits and Professor Richards on beneficiation of the ore were favorable, no further work appears to have been done and no reason given for cessation of work.

During my exploration of the north and south shores of Newfoundland in 1898 and 1899, my attention was called to these properties but I did not visit them because the international treaty of Utrecht granted the French a narrow strip of land along the entire west and a portion of the north coast of Newfoundland, and although this treaty did not grant the land in fee, the French would not permit trespassing or contact with the ocean for shipping purposes. One needs only to examine the map of Newfoundland to note that the railway which was originally surveyed to terminate in St. George Bay was carried several miles south to Port au Basque for its terminus. It is my opinion that Mr. Maynard's work ceased when he learned of the French claim to the coast line.

Comparatively little has been published on Newfoundland other than the early geological reports by Sir Alexander Murray and Mr. Howley. So far as I know no comprehensive report has been made on the prospective possibilities of the chrome deposits; they may or may not be of value.

Occurrence of Quicksilver Orebodies

By C. N. Schuette, San Francisco, Calif.

(New York Meeting, February, 1930)

The material presented in this paper has been gathered by the writer during a long and varied experience on matters pertaining to the quicksilver industry. During the past 18 years he has visited practically all of the quicksilver mines of the United States and has had occasion to review the literature on quicksilver mining the world over.

In a previous publication[1] the conclusion was expressed "that the greatest opportunity for increasing the economy in quicksilver production lies in giving more attention to the geology of the deposits . . . " The present paper is a contribution toward that end.

Due to a lack of correlation of the observed facts, a great deal of confusion exists in the views of various writers concerning the mode of occurrence of quicksilver ores. This paper attempts to show that certain common factors underlie the formation of all quicksilver orebodies. It attempts to present a framework on which to marshall the observed facts and thus make clear the interrelation of various observations which at first sight appear to be contradictory.

The material presented is fragmentary in many respects and it is hoped that readers familiar with the deposits mentioned may supply the deficiency. The reader is asked to refer to the publications cited for maps and additional information on the individual orebodies, as this review, in covering so much territory, must necessarily confine itself to a bare outline of the facts concerning each occurrence.

The writer hopes that this paper will elicit comment and criticism from the many competent men interested in the quicksilver industry and to all of them he would express his indebtedness for the many courtesies that they have extended to him at various times in the past.

Theory of Primary Concentration

The mode of occurrence of any given orebody presents an individual problem to the miner engaged in exploiting it. No two orebodies are exactly alike in their geological relationships, therefore an understanding, in detail, of the geology is necessary if the orebody is to be exploited in the most economic manner.

[1] L. H. Duschak and C. N. Schuette: The Metallurgy of Quicksilver. U. S. Bureau of Mines *Bull.* 222 (1925).

Aside from the characteristics of the individual orebodies, certain types exhibited group characteristics. These may relate to structure, outcrops, age of the enclosing rocks or other geological factors. Such group characteristics are the determinants used in prospecting. For example, oil fields are located by a study of structure; the "iron hat" may indicate copper deposits, and coal in certain localities is sought in rocks of Carboniferous age.

A study of quicksilver orebodies in the United States, supplemented by a thorough review of the literature relating to quicksilver mines the world over, leads to the conclusion that certain geological group characteristics of great importance in prospecting, developing and mining are shown by quicksilver ore deposits.

In general ore deposits are thought of as being either primary deposits or deposits derived from primary deposits by a process of secondary enrichment.

Many primary deposits of mineral are known that are not orebodies. Some primary deposits of mineral, which in themselves were of too low a grade to form orebodies, have been the source of orebodies formed by a secondary enrichment of the primary mineralization.

This paper treats of a third group of ore deposits, which formed orebodies by primary enrichment or a concentration of the primary mineralization during deposition, and thus differs from the orebodies of most other metals.

Ore deposits formed by primary concentration have distinct characteristics that aid in finding them. They often form high-grade orebodies. They do not necessarily outcrop and their discovery in the past has been fortuitous rather than the result of intelligently directed prospecting. For this reason relatively more deposits of this type probably remain to be discovered than of types that outcrop prominently.

Quicksilver ore deposits are practically all formed by primary concentration. It must be understood by the reader that this article is written from the prospectors' and miners' viewpoint and that it relates to orebodies only and not to uncommercial mineral occurrences, however interesting these may be geologically. The theory of the genesis of quicksilver orebodies by primary concentration is outlined first and is then elaborated by examples which show that the largest and richest quicksilver orebodies most nearly fulfill the precepts of the theory.

The theory as outlined evolved slowly through the course of years with many modifications due to new observations and attempts to correlate published descriptions relating to the various deposits. There are many loose ends to be tucked in by future investigations and gaps to be filled by accumulating knowledge. Its presentation is an attempt to correlate past studies of individual deposits by various geologists and to

form a base from which to extend future explorations into the field of the genesis of quicksilver ores.

Briefly stated, the theory of ore deposits formed by primary concentration, a classification which includes practically all quicksilver ore deposits, is as follows:

1. The source of the ore is a deep-seated igneous rock magma.

2. The ore minerals are carried to the point of deposition by hot alkaline solutions ascending through fissures in the rock.

3. The ascending mineral-bearing solutions are directed and limited or even dammed at some point in their upward course by a relatively impervious rock.

4. Precipitation of the ore minerals is caused by cooling and dilution of the mineral-bearing solutions, by loss of pressure or by precipitating agents such as organic matter or gaseous reagents.

5. The orebody forms in any pervious rock or in the interstitial spaces of any broken rock mass or in other voids underlying the relatively impervious cap rock.

6. The forming of the orebody is due to the concentration of the ore mineral in the trap formed by the relatively impervious rock. This trap structure has directed and limited the upward flow of the mineralizing solutions to the porous rock mass below.

7. The ore minerals are predominantly primary minerals, secondary minerals being rare and of little importance as ore.

The inevitable exceptions to these somewhat categorical precepts will be noted and discussed in the description of the individual orebodies. The theory has been presented in this form for the purpose of facilitating comparisons.

Many geologists[2–7] have noted that quicksilver deposits the world over show a close association with volcanism and major fractures in the crust of the earth. In regions where the deposits occur in older rocks a reconstruction of the geological structure as it probably was before erosion changed the aspect is necessary to make this clear. In younger regions where volcanism is active or was active until recent times the relationship is evident and at some points, such as Sulphur Bank and

[2] G. F. Becker: Geology of the Quicksilver Deposits of the Pacific Slope. U. S. Geol. Survey *Monograph* 13, (1888) 452–453.

[3] J. A. Udden: Structural Relations of Quicksilver Deposits. *Min. World* (May 13, 1911) 975.

[4] J. A. Veatch: The Mercury Deposits of the Pacific Coast. *Min. & Eng. World* (March 28, 1914) 591.

[5] W. H. Emmons: The Enrichment of Ore Deposits. U. S. Geol. Survey *Bull.* 625 (1917) 398.

[6] F. L. Ransome: Quicksilver. U. S. Geol. Survey *Min. Res. of U. S.*, 1917, Pt. I, 386.

[7] F. R. Tegengren: The Quicksilver Deposits of China. Geol. Survey of China *Bull.* 2 (October, 1920) 5–6.

Steamboat Springs, cinnabar is probably being deposited at the present time.

Thus it seems reasonable to conclude that the source of the ore is an igneous rock magma when regions in which there is evidence of the existence of such magma in an active state are identical with regions where the ore is found and is still being deposited.

Quicksilver has an appreciable vapor pressure even at ordinary temperatures and must be one of the most volatile metal constituents of rock magmas. Suppose an active magma underlies a volcanic region that is undergoing fracturing. It is reasonable to suppose that any fracture penetrating to the magma or starting from the magma would release water, quicksilver and sulfur. These would be the first to escape through the fissure in such condition, probably liquid, as the temperature and pressure would indicate.

Fissures of this kind extending to the surface would discharge the ascending magmatic constituents as steam and vapor, or if the cooling action of the walls or of admixed surface waters were great enough, as hot springs. The magmatic ascending solutions might also be intercepted and diluted by surface waters to such an extent as to lose their identity. In all of these supposed cases the formation of a quicksilver orebody would be extremely doubtful, though cinnabar might be deposited in small quantity on the surface. This would be of interest to a geologist but not to the miner. The condition necessary for forming an orebody is a trapping of the mineral-bearing solutions so that the contained mineral will be concentrated.

Once the ascending magmatic solutions have been arrested and confined, various factors may contribute to the deposition of the contained mineral. Cooling, dilution, loss of pressure, organic matter and dissolved gases are all not only possible but very probable factors in this process. Their efficacy in causing precipitation has been proved in the laboratory[8–10] and proof of their action is found in the ore deposits also.

Christy[11] first advanced the theory that the ascending solutions were alkaline, Becker[12] agreed and Allen and Crenshaw[13] confirm it by stating: "The evidence for the geologic view that cinnabar is a product of alkaline solutions is convincing." Whether the constituents of these solutions other than the quicksilver, water and probably sulfur

[8] S. B. Christy: On the Genesis of Cinnabar Deposits. *Amer. Jnl. Sci.* [3] (1879) **17**, 453.

[9] F. W. Clarke: The Data of Geochemistry. U. S. Geol. Survey *Bull.* 616, (1916) 664.

[10] F. F. Grout: On the Behavior of Cold Acid Sulphate Solutions of Copper, Silver, and Gold with Alkaline Extracts of Metallic Sulfides. *Econ. Geol.* (1913) **8**, 417.

[11] S. B. Christy: *Op. cit.*

[12] G. F. Becker: *Op. cit.*, Chap. 15.

[13] Allen and Crenshaw: The Sulfides of Zinc, Cadmium and Mercury. *Amer. Jnl. Sci.* [4] (1912) **34**, 380.

are of magmatic origin or whether they are taken from the rocks through which the magmatic emanations rise is an open question.

In order to illustrate the next precepts of the theory, assume that an active magma has intruded and uplifted a series of bedded sedimentaries. The sedimentaries affected are limestone, sandstone and shale in ascending order, as shown in Fig. 1. The bending stress during the uplift has

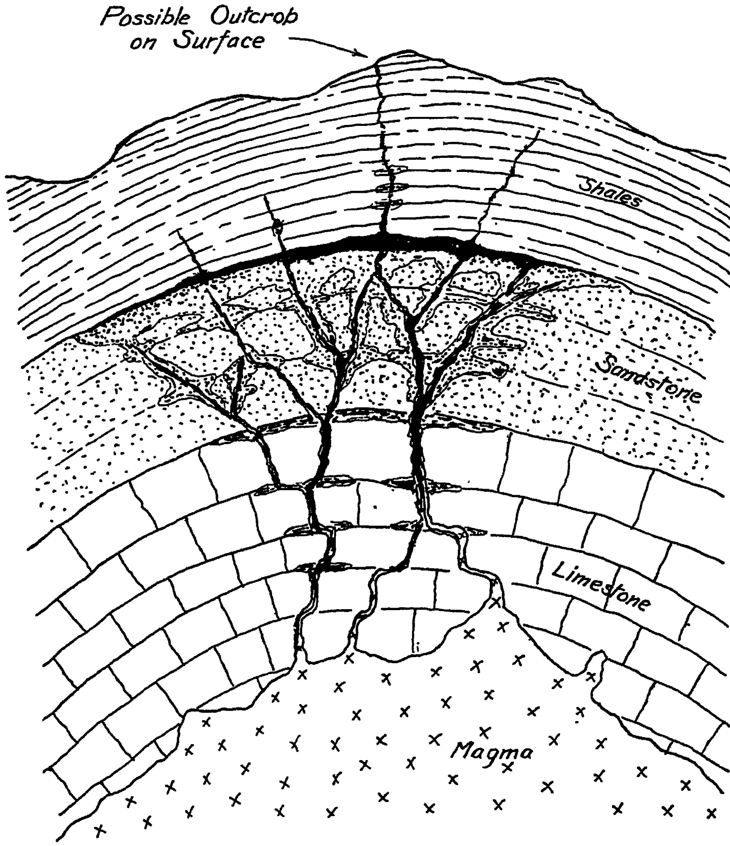

Fig. 1.—Intrusion and uplift of bedded sedimentaries by an active magma.

caused fracturing in the back of the anticline so formed. In the hard limestone the fractures would be open with sharp walls. In the overlying sandstone they would be more or less open, depending on the degree of cementation and on bedding planes. In the softer shales above the sandstone the fissures probably would seal themselves or perhaps would not be formed on account of the yielding nature of the shales. The ascending mineralizing solutions, being practically sealed off by the impervious shale, would spread through the porous sandstone and form a mineral deposit. The concentration of this primary mineralization

would be greater near the shale contact and would spread irregularly through the sandstone in accordance with variations of porosity. If enough mineral were brought up, the feeder fissures in the open limestone would also be filled with mineral. The outcrop, if any, would be a thin irregular stringer through a possible fissure reaching to the surface through the shale. The arrest of the magmatic solutions and their confinement to a restricted area gives the proper conditions for concentration by cumulative deposition over a period of time.

Ore formed under these conditions must necessarily be primary, as secondary minerals ordinarily could form only after the orebody had been exposed to surface agencies by erosion.

The significance of this mode of occurrence to the prospector and miner is apparent in Fig. 1. If the presence of the sandstone were not known the sketchy outcrop would not seem attractive. If the structural relations could be determined by a geologist, a drilling campaign would result in discoveries meriting development. The mining based on this development would be concerned with the exploitation of the large spread-out orebody at the shale sandstone contact, the various fissure orebodies and the impregnations from these into the sandstone and lime. The dominant realization governing these activities would be that the ore is found in positions that were favorable to the concentration of the primary mineralization.

The primary mineral of practically all quicksilver orebodies is cinnabar. There are some 25 minerals containing quicksilver but most of them, while interesting mineralogically and geologically, are of no interest in this discussion, which confines itself to a consideration of ore occurrences. Two quicksilver minerals beside cinnabar are often referred to as ores of quicksilver. The first of these is native quicksilver. The occurrence of "native"[14] is generally confined to the vicinity of the surface, and represents a secondary alteration product from cinnabar due to acid waters and organic material found at the surface. Cinnabar may be soluble in acid solutions[15] and native mercury possibly is precipitated from such a solution by the combined action of H_2S and organic matter. While small amounts of native metal are found in many quicksilver mines, there are few deposits where it occurs in sufficient abundance to be classed as an ore.

Metacinnabar is often, but I believe erroneously, referred to as an ore of quicksilver. Becker,[16] speaking of the Redington mine, says, "In the upper and richer portion of this mine a large part of the mercury

[14] In the parlance of the quicksilver industry "native" is used as a noun to denote metallic quicksilver occurring as such in nature.

[15] T. M. Broderick: Some Experiments Bearing on the Secondary Enrichment of Mercury Deposits. *Econ. Geol.* (1916) **11**, 645.

[16] G. F. Becker: *Op. cit.*, 285.

was in the form of metacinnabarite," and again, "So entirely had the accessible portions of the upper levels of the Redington mine been worked out at the time of my visit that I was unable to find any of this ore in place." This statement does not sound reasonable if metacinnabar really constituted the greater part of the *ore*. There is ore in place in the reopened upper workings of this mine today but very little of it, if any, is metacinnabar. It is possible, of course, that an orebody of cinnabar exposed to surface conditions by erosion would be attacked by acid waters and these same acid waters acting on iron sulfides could generate H_2S to reprecipitate the black HgS. It is doubtful however that the entire orebody would be so changed. More probably only the surface of the cinnabar would be coated with the black sulfide. Specimens of metacinnabar are rare and most of them are coatings, as indicated. The mine operator is rather careless of his definitions and many specimens from carloads of so-called metacinnabar going to the furnaces proved on careful testing to be cinnabar coated with a black substance, which in some cases was the black sulfide but more often was carbonaceous material, a black iron stain or manganese dioxide. While there is no doubt regarding the occurrence of metacinnabar in quicksilver orebodies, it is hardly to be considered as an *ore* of quicksilver, the statements of many assiduous promoters to the contrary notwithstanding.

All of the quicksilver minerals including cinnabar may be present in an orebody as secondary minerals. Generally however, the unmistakable primary cinnabar predominates so greatly as to exclude the secondary minerals from consideration as ore. Broderick[17] has shown how most of them may be formed in nature. There is in quicksilver mining no such secondary enrichment as in copper or silver mining. The reason for this lies in the fact that the precipitating agents of the dissolved quicksilver ore are present in the same horizon as the solution agents and reprecipitation takes place *in situ* without the migratory movement that results in enrichments. For the practical purpose of mining, then, the orebodies of quicksilver can be regarded as primary orebodies. Metallurgically also any secondary minerals present no problem, as they yield to the same treatment as the primary cinnabar.

Just as with the occurrence of native and metacinnabar, so has the occurrence of bitumen[18,19] in some quicksilver mines been greatly exaggerated. It is rated as being of great significance by assiduous textbook writers copying one from the other. Actually its presence may affect the mining cost a trifle by necessitating sufficient ventilation to prevent accumulations and consequent explosions of methane gas.

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[17] T. M. Broderick: *Op. cit.*, 651.

[18] The Birth of Quicksilver Ores. Ed., *Eng. & Min. Jnl.* (1922) **114,** 45.

[19] C. N. Schuette: The Birth of Quicksilver Ores. Ed., *Eng. & Min. Jnl.* (1922) **114,** 228.

Bituminous matter is not nearly so common in quicksilver mines as one would believe from the exaggerated accounts of its occurrence. It is merely an attendant occurrence and of subordinate geological importance. Where it is found, its origin is generally quite clear and if it were not present its absence would be a phenomenon. Generally, where bitumen is found in a quicksilver mine the underlying rocks are sedimentaries containing organic matter. Heat developed in these rocks by proximity of the mineralizing magma, by the strain attendant on fissuring or by the hot ascending ore-bearing solutions, no doubt initiates distillation and stimulates migration or organic bituminous matter. This ascends in the same fissure with the ore-bearing solutions and is trapped and deposited in the same horizon as the cinnabar. This furnishes a much more probable explanation of the occurrence than the often advocated magmatic origin for the bituminous matter as found.

In reviewing the literature on quicksilver deposits one is struck by the fact that most writers are confused by being unable to correlate their observations in different districts despite many similarities. Most observers agree that quicksilver deposits are exceedingly irregular, are generally close to the surface, that they occur in rock of any age or kind and that they are generally associated with volcanism or at least solfataric action. They agree that the orebodies are not true veins and call them beds, stockworks, breccia fillings and impregnations. One is also struck by the imperfect descriptions of the orebody and its general characteristics. Many writers do not even mention the hanging or footwall and their characteristics or the grade of the ore.

Paragraphs are written on the country rock, analyses are given and the age of the rocks is guessed. Pages are devoted to the possible significance of a rare quicksilver mineral but the description of the orebody itself is woefully neglected.

The features of quicksilver orebodies that are of greatest interest to the miner and that should be fully described are the dip and strike of the structure, the hanging wall, the footwall, the gradation of the ore value from the hanging to the footwall, the porosity of the ore-bearing rock and the location of feeder fissures.

The proper trap structure must be present before an orebody can be formed. The hanging wall is generally a clay gouge or other relatively impervious rock which forms a sharp dividing line between ore and waste. The footwall, if an impervious rock, may determine the other limit of the orebody but more often there is no definite footwall because the ore grades off from the hanging to a "commercial" footwall depending on the price of quicksilver.

The porosity of the rock generally determines the grade of the ore formed; for example, a coarse sandstone impregnated with cinnabar forms higher grade ore than a fine-grained shale with less interstitial space.

Feeder fissures are important, as branches from them may lead to new orebodies and their strike and dip in relation to the structure is sometimes of great significance. There are, of course, many other significant features, some of which will be touched upon in the following description of the individual mines, but the above are the essential features of every quicksilver orebody.

In order to show that quicksilver mines the world over have certain fundamental characteristics in common, and that they govern to a large extent not only the mining practice but also the cost of mining, the following chapters are devoted to a description of the different types of quicksilver mines in various quicksilver-mining districts.

Quicksilver Deposits in California

New Almaden

The New Almaden quicksilver mine lies in Santa Clara County, California, about 12 miles east of south from San José. It has produced some $75,000,000 worth of quicksilver in the past 100 years and in point of total production outranks all other quicksilver mines in the United States. The orebodies lie in a low range of hills which has a northwest and southeast trend. Mine Hill, 1755 ft. high, dominates this range of hills.

The country rock of the region consists of sandstones and shales of the Franciscan series. The Franciscan rocks are probably of Jurassic age. Along the trend of Mine Ridge from the New Almaden to the Guadalupe mine, these sandstones and shales were at some time intruded by peridotites. Alteration of this peridotite to serpentine followed. The intrusion itself and perhaps also the volume change accompanying the serpentinization caused intense fracturing of the rocks along the contact.

With these elementary facts in mind, consider the occurrence of the ore deposits. The footwall of the orebodies, *i. e.*, the underlying rocks, is at all points serpentine. The hanging wall is at all points a clay gouge locally known as *alta,*[20] and above this alta lie the Franciscan sandstones and shales.

That a deep-seated igneous rock magma underlay this region is evident by the presence of the serpentine as well as an extrusion of rhyolite to the northwest of the mine and more generally by exposures of the Montara granite far to the south. Thus a deep-seated magma is indicated as the original source and this view is strengthened by many mineral springs on the property and the occurrence of diorite dikes in the mine. Fissuring along planes of weakness reaching the point of ore deposition is indicated by the fractured zone on the serpentine-sand-

[20] In the early days of quicksilver mining in California, the term *alta* was in common use to denote the usual clay-gouge hanging wall of quicksilver deposits.

stone contact and by the plane of weakness indicated by the dike of rhyolite. Either the original peridotite intrusion or the change in volume during the serpentinization of the peridotite or both caused movement and consequent attrition between the serpentine and the overlying rocks. This movement fractured the contact zone and one of the attrition products was the clay-gouge alta at or near the original contact. Just how near the surface this action took place is problematical, though the serpentinization itself as well as the gangue minerals quartz, calcite, dolomite, opal and pyrite, place the occurrence in the epizone in accord with the theory of hydrothermal ore deposition. The formation of a fractured or brecciated zone capped by an impervious clay gouge provided the conditions necessary for the formation of an orebody by a concentration of the primary mineralization.

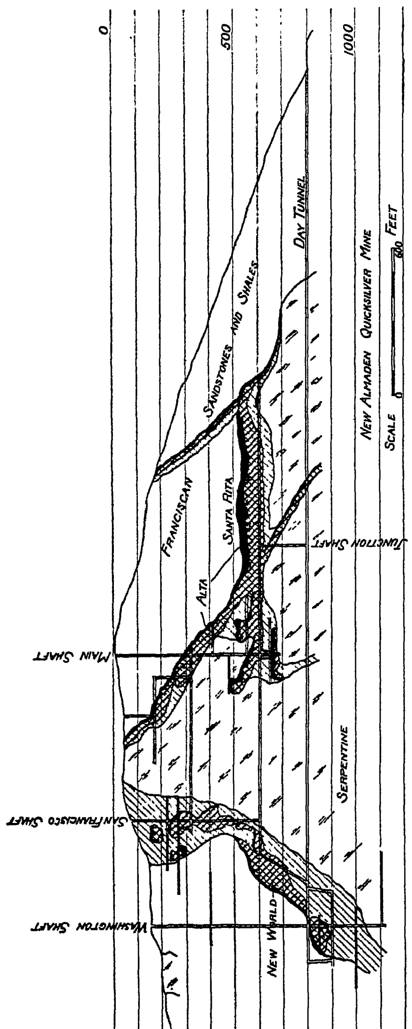

FIG. 2.—GENERALIZED SECTION THROUGH NEW ALMADEN MINE ALONG THE DAY TUNNEL.

The hot mineralizing solutions ascending through the fractured contact zone along the plane of structural weakness were directed and limited by the impervious altas to the zone of fractured rock under them. Here the trapped solutions deposited their burden of mineral by precipitation from one or more of the causes enumerated above.

Fig. 2 shows a generalized section through the New Almaden mine along the course of the Day tunnel. The shaded areas indicate the fractured zone along the contact through which the solutions rose, while the crosshatched parts indicate the orebodies. The altas are shown in black and their influence on the formation of the orebodies is clearly evident. A flattening of the alta generally indicated more or

higher grade ore. The following paragraph will complete the picture of the orebodies as presented on the section.

The Randol shaft lies some 1200 ft. behind the section shown, about halfway between the mouth of Day tunnel and Junction shaft. Orebodies on each side of the Randol shaft extended upwards from below the 2000-ft. level north and west of the shaft and were continuous with the large flat Santa Rita orebody shown under the heavy alta. These orebodies are well illustrated by Becker.[21] They were explored and exploited through the Randol, Buena Vista and Santa Isabel shafts. In front of the section, Fig. 2, and opposite the Junction shaft at distances of 900 and 1400 ft. are the Harry and Cora Blanca shafts from which orebodies rising toward Mine Hill were mined.

Thus in a very general way the orebodies of New Almaden were distributed on an irregular hump of serpentine apexing in Mine Hill. They were deposited in the voids of a brecciated zone under an umbrella of impervious attrition gouge which trapped the mineralizing solutions.

The mining history of this paragon of orebodies is interesting and instructive in retrospect. The ore was first discovered at the outcrop to the left of the hilltop in Fig. 2 and was worked through short tunnels and shallow shafts. In 1851 the main tunnel (300-ft. level) was driven under the top of Mine Hill and the ore from there to the surface was extracted. Next the main shaft was sunk from this tunnel to the 600-ft. level. Water and ventilation presented their usual problem and in 1857 the Day tunnel was started to drain the 600-ft. level and take out ore. In 1864 connection was established through the Junction shaft.

When the Quicksilver Mining Co. took over the property in 1863 the ore had been worked out down to and including the Ardilla stope on the 500-ft. level. There was little ore in sight, no other orebodies were known and the outlook was doubtful. The average grade of the ore mined up to this time had decreased from 37 per cent. in 1850 to 18 per cent. in 1863. This was considered to be very discouraging.

A thin stringer of ore on the hanging-wall alta was followed from the Ardilla in 1865 and led to the North Ardilla stope, which supplied ore to the reduction plant for some seven months.

As this stope petered out the hanging-wall stringer was followed down and another orebody was found. This find developed into the immense flat-lying Santa Rita orebodies indicated on the section, which yielded[22] some 25,300 tons of better than 10 per cent. ore, or about 70,000 flasks of quicksilver.

As these orebodies were followed to the north and west (behind the section) they became smaller and dipped down, with one exception.

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[21] G. F. Becker: *Op. cit.*, atlas sheets X, XI, XII.

[22] J. B. Randol: Report on Mineral Industries in United States—Quicksilver, 11th Census (1890) 213.

This exception is shown on the section, Fig. 2, at the right hand end of the flat-lying orebody. This fissure in the Franciscan strata represents the shear zone of a fault fissure due to an apparent change in direction of the movement along the contact. Tunnels were driven to the upper parts of this orebody and smaller quantities of high-grade ore were mined from it between 1864 and 1872.

Here was a much shorter lead to the rich Santa Rita orebodies, which, probably because of an indifferent outcrop, either was not found or was not considered important enough to merit development.

The orebodies dipping to the north and west from the Santa Rita stope were irregular and discontinuous but the altas were followed down to the 900-ft. level by 1873. Now, as 10 years before, the outlook for the mine was considered doubtful. The grade of the ore sent to the reduction plant had decreased from 18 per cent. to 5 per cent. in 10 years. The situation was considered to be desperate.

Since the altas dipped to the north and west and since ore if present was always under the altas, shafts were sunk in the territory north and west of the Santa Rita orebodies. The Randol shaft was started in 1871, the Santa Isabel in 1877, the Buena Vista in 1882, the St. George in 1887 and later still the Church and America shafts were sunk. The Randol shaft, as it proved later, was located in the best possible place, but unfortunately it had but a single hoisting compartment and one pump and ladderway. It was this peculiar combination of circumstances that caused the Randol shaft to become the Old Man of the Sea of the Almaden mine.[23] The trouble began when water was struck 100 ft. below the Day tunnel level. It took all of the power of the hoist engine to pump, so that hoisting had to be suspended when pumping. Two hundred feet below the Day tunnel the same thing happened again but this time the pumps were drowned. This trouble resulted in the release of James Pearce, who was held responsible for the inadequacies of the shaft. Captain Grey succeeded him and, though engineers condemned the shaft as too small, Grey, a conservative old timer, prevailed and it was sunk 600 ft. more in 100-ft. lifts with a bucket and windlass, under almost impossible conditions.

Finally, the 6-in. pumps had reached their limit. Meanwhile the Hacienda tunnel project had been abandoned, the Cora Blanca was down, no ore had been found in the Grey shaft, the New World orebody was petering out and the furnace plant called for 400 tons per day.

The Randol shaft could hoist only 300 tons when working continuously, and it needed deepening. One-half million tons of ore were

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[23] Much of the following history is taken from an unpublished manuscript by A. C. Innes, entitled "J. B. Randol's Administration of the New Almaden Mines from 1870 to 1889."

available in the mine but could not be brought out fast enough, so the Santa Isabel shaft was sunk in 1878 to drain the mine and handle ore. This three-compartment shaft proved some new ground and handled the water but was of little help in handling ore, so that by 1882 about 800,000 tons of broken ore had accumulated in the Randol workings, which could only be hoisted at the rate of 300 tons per day. Two expensive shafts were now doing the work of one with no increase of output. Any temporary shut-down of the Randol shaft affected production just that much, and with the stopes full of broken ore the furnaces at times were fed on dump material to keep them in operation. The new shaft and the Garfield shaft necessitated the expenditures of large sums for the sinking and the attendant road building and housing facilities. The Randol shaft was deepened and ore development from it continued. The limit of the pumps in the Santa Isabel was reached and another shaft, the Buena Vista, was sunk to the 2300-ft. level. It was connected to the Randol workings through long crosscut tunnels on the 2100 and had 10-in. pumps. While it had a hoisting capacity of 1000 tons per day, it never raised a ton of ore and served for pumping only. Meanwhile, hoisting 300 tons per day through the Randol became even more expensive as depth was gained.

The Garfield, renamed the Washington shaft, was unproductive.

The ground between the Enriquita and Mine Hill had long been thought of favorably. Here from 1885 to 1887, on Bull Run, the America shaft was sunk in a desperate effort to find ore reserves to offset the diminishing orebodies near the Randol shaft. A drift from the 600-ft. level Santa Isabel was driven under Bull Run to drain the country and the America shaft was started. Owing to water and gas, this proved to be so expensive that the expenditures of the mine greatly exceeded income and work was abandoned after sinking 800 ft. The Randol shaft produced its quota for some years longer but all developnent north of Mine Hill had come to a standstill and Randol left the mine soon after.

The annual tonnage from 1873 to 1893 was more than double that of the 10 years preceding while the grade of the ore declined steadily until in 1895 it definitely dropped below 1 per cent.

The region south of the main shaft was explored by tunnels on the 300-ft. level, starting in 1864. The orebodies here were irregular under altas erratically situated. In 1869 the San Francisco shaft, Fig. 2, was sunk from the 300 to 600-ft. level and important orebodies were stoped. In 1874 the New World, a high-grade orebody trapped under a flat alta, was discovered from the 600-ft. level and followed down to the 800-ft. or Day tunnel level. The Washington shaft was completed in 1882 to exploit this orebody and prospect for extension in depth. The orebody thinned out and only reached to the 850-ft. level. Prospecting here was abandoned in 1887.

The Cora Blanca orebody was found in 1864 and the ore near the surface was worked out. In 1873 the shaft was deepened and new ore was developed down to the 800-ft. level. The Grey shaft farther southeast, sunk in 1876, failed to find ore below the 800-ft. level and the workings were abandoned in 1879. After Randol's time the Harry shaft between Mine Hill and the Cora Blanca opened a large body of ore which proved to be continuous with the other orebodies in Mine Hill.

This history is given in some detail, because it is interesting and to bring out certain facts that are also true in a general way of many other quicksilver mines the world over. The outcrops were relatively small and insignificant. The immensity of the orebodies was not even suspected at the beginning of operations. Had it not had a strong aggressive management, the mine might have been abandoned several times as being worked out. The extensions of the orebody were found by following the alta, which was recognized as the most reliable guide in prospecting.

Had the nature of the ore deposit been understood and had churn drilling been used in prospecting, the extent, scope and nature of the orebody would have been revealed. A much more economical method of mining and handling the ore thus made possible would have greatly enhanced the economic return of the mine. Dividends commensurate with the value of the output could have been paid, instead of practically nothing, as was done. The expense of shafts, which served merely to keep the most inefficient one in operation and added not one whit to production while they did add to cost, could have been saved. The disastrous Bull Run or America mine project would never have been started, as work through preliminary tunnels and shafts from 1863 to 1874 had disclosed that the ore-bearing fissures became narrow and almost vertical in depth; certainly not a structure that would be conducive to a great concentration of the primary mineralization.

There are numerous cinnabar prospects between Mine Hill and the Guadalupe mine 4 miles to the northwest. Some of these, such as the Enriquita, were developed and this particular one yielded some 9000 flasks. One, The Senator, was the scene of the last mining activity of New Almaden at the time of this writing.

A large territory remains to be explored for favorable structures. Becker's geological map of the New Almaden district is in need of revision, as many inaccuracies have been observed. His was a splendid pioneer effort in a new and geologically unexplored region but since that time many new facts have been brought out that must now be taken into consideration.

In general, then, the dip and strike of the New Almaden orebodies followed that of the serpentine-sandstone contact. The hanging wall was sharply defined by the clay-gouge alta. The footwall was either commercial where the ore graded off from the hanging or was the limit

of brecciation in the serpentine. The voids in which the ore was deposited was the interstitial space of a brecciated zone. The feeder fissures seem to have been the fracture zones themselves, though a few feed fissures were followed down into the underlying serpentine. The trap structure of the Santa Rita ore chambers was very nearly ideal, being a flat-lying breccia capped by a heavy impervious gouge.

New Idria

This mine, the second largest producer in the United States, as measured by total production to date, lies in San Benito County, California, about 70 miles southeast of Hollister. It has been in continuous operation since 1850.

Like New Almaden, the mine lies near the contact between serpentine and Franciscan sandstones and shales. This body of serpentine as exposed trends northwest and southeast, is some 15 miles long and 3 to 4 miles wide. The peridotite from which the serpentine was derived is intrusive into the Franciscan strata. Apparently unconformably on the Franciscan lies the Panoche formation of upper Cretaceous age. Geologic maps of the New Idria area are given by Becker[24] and by Bradley.[25] Comparison of the two maps indicates clearly the many additions to geological knowledge that have been made since Becker's report.

Forstner[26] in his description of the quicksilver district of southern San Benito County shows that the dip of the strata south of the serpentine area is 70° toward the south while Bradley's map and Becker's description give the dips north of the area as some 60° north. A north and south section is given in Fig. 3*b*, which suggests a flexing of the overlying rocks by the serpentine. The New Idria mine lies on the northern slope of the mountains comprising the serpentine area. The orebody outcropped on this slope at an elevation of 3750 ft. The strike of the croppings is about east and west and the dip of the deposit lies between 60° and 65° to the south. The hanging wall is an attrition gouge caused by movement between the serpentine and the Franciscan rocks. The footwall of the mineralized zone is another gouge caused apparently by movement between the Franciscan and the Panoche strata. The Franciscan rocks are shattered and constitute the receptacle rock in which the orebodies were formed. The mine is being worked on the 1400-ft. level through a shaft sunk from No. 10, the main haulage level. The dip and strike of the Panoche formation is revealed in entering the mine through this tunnel. As the ore-bearing zone is approached, the strati-

[24] G. F. Becker: *Op. cit.*, atlas sheet VI.

[25] W. W. Bradley: Quicksilver Resources of California. California State Mining Bureau *Bull.* 78, Plate XII.

[26] W. Forstner: The Quicksilver Resources of California. California State Mining Bureau *Bull.* 27 (1903).

fication becomes irregular and disturbed, as indicated in Fig. 3*b*. Fig. 3*a* shows the position of the orebodies in relation to the dip and strike.

a

b

c

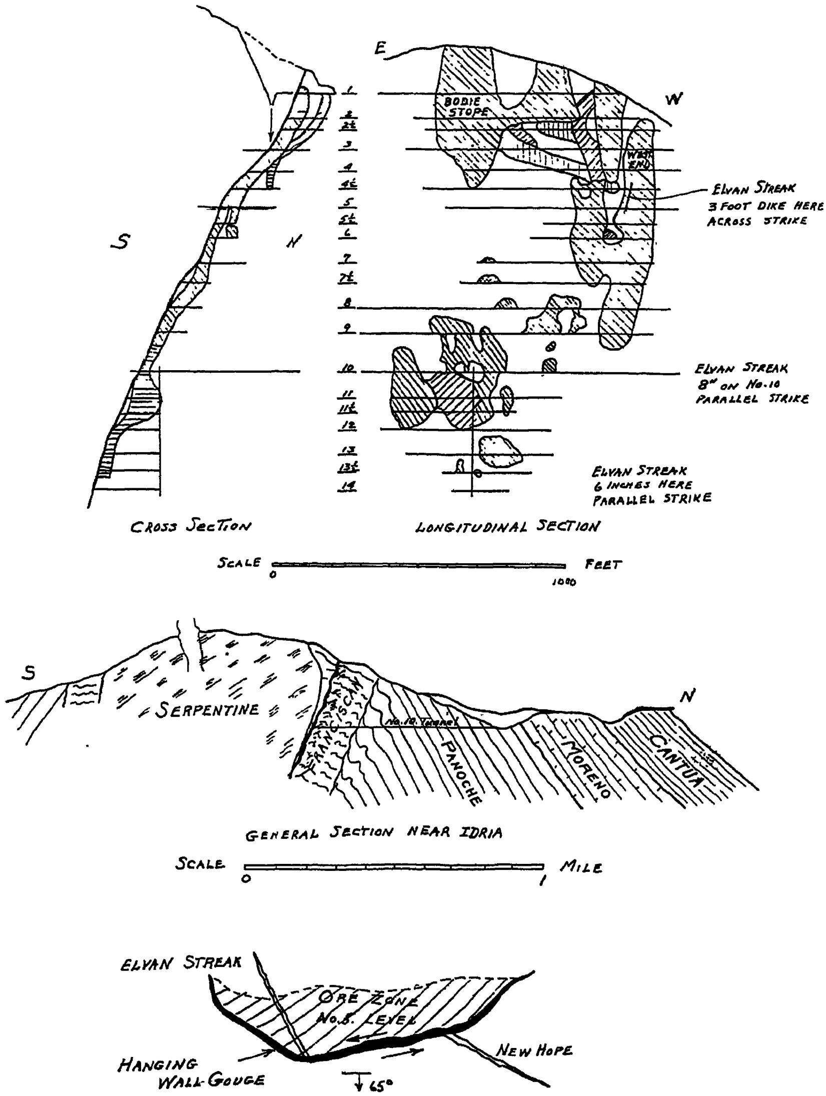

FIG. 3.—NEW IDRIA MINE.

a. Position of orebodies in relation to dip and strike.
b. North and south section.
c. Relation of hanging-wall gouge and Elvan streak.

This illustration is the work of E. B. Dane, Jr., the geologist of the New Idria company.

An inspection of the level maps of the mine reveals that a horizontal section of the hanging-wall gouge gives a sickle-shaped figure, the horns of which curve north. Since the deposit dips south, this gives to the gouge the general shape of a gutter pipe standing at a high angle, bottom side up. This has been illustrated by Forstner[27] and Bradley.[28] Two other features are mentioned in all publications concerning the New Idria mine—the New Hope and the Elvan streak.

The New Hope is a fissure in the hanging wall of the orebody; that is, it lies above the hanging-wall gouge and reaches only to this gouge. It was worked only on the upper levels; it is said to have been rich and the ore is said to have been metacinnabar. The Elvan streak cuts across the ore zone under the hanging-wall gouge. The relation of the two on the fifth level is shown in Fig. 3*c*. Becker[29] says of the Elvan streak, "This is a misnomer, for Elvan is quartz porphyry while at New Idria neither this nor any other eruptive rock occurs. The Elvan streak is for long distances a clean-cut fissure filled with decomposed attrition products which are impregnated with cinnabar.

"This vein strikes in about the same direction as the New Hope and near it at one point were found some tons of metacinnabar. To the southeast it is cut off by a clay seam."

Whoever named the Elvan streak was not as far wrong as Becker assumed as the "decomposed attrition product" is an altered intrusive and the Elvan streak persists to the lowest levels, although here its position in the fractured zone conforms with the strike of the ore deposit.

The story of the ore occurrence at New Idria is probably as follows. The pressure due to volume increase attending serpentinization of the peridotite intrusion developed a plane of weakness along the Franciscan-Panoche contact as evidenced by the crumpled condition of these rocks near the contact. In this zone of weakness a rift developed through which an intrusive pushed its way to or near to the surface. Below what is now the No. 5 level of the mine, the strike of this rift was east and west, while higher up it was more northwest and southeast. As the intrusive cooled and shrank the mineralizing solutions rose through the rift. Ore was being deposited in at least the upper part of the rift when another movement took place. The northern strata moved west and the southern strata east, forming the attrition gouge of the present hanging wall and displacing the rift into two parts, which are the New Hope and Elvan streak of today. This would fix the movement which formed the curved hanging-wall gouge as the latest movement. This view conforms with a fault exposed on No. 10 level, which offsets the receptacle rock in which the ore was formed but did not displace the hanging-wall gouge, and

[27] W. Forstner: *Op. cit.*, 140.
[28] W. W. Bradley: *Op. cit.*, 110.
[29] G. F. Becker: *Op. cit.*, 302.

therefore indicates that this gouge was formed by the latest fault movement.

Here as at New Almaden there is evidence of an active magma as the source of the ore and evidence of planes of weakness or fissures reaching to the magma in the form of a dike rock in the Elvan streak. The mineralizing solutions ascended under an inverted trough of impervious gouge, which confined them to the brecciated rock below. This restricted the deposition of cinnabar to this zone and so caused a concentration of the primary mineralization. The Elvan streak was the fissure through which the solutions ascended and the intrusive dike rock, probably an offshoot from the mineralizing magma itself, was altered by these solutions. New Hope represents a displaced part of the Elvan streak and some of the primary cinnabar in it was altered to metacinnabar by acid surface waters.

Here in the New Idria district is a second large area meriting a detailed study for favorable structures. Since Becker's time no geologic work has been done in this region except that by Pack and Anderson, which had to do with oil and only touched New Idria incidentally. Oil is plentiful and many financially able companies keep staffs of geologists in the field. Quicksilver is needed and few quicksilver-mining companies can afford to hire geologists for regional mapping. Detailed mapping of this region by government geologists might prove well worth while from a standpoint of fostering our natural resources.

The serpentine-sandstone contact is evidently a plane of weakness along which quicksilver-bearing solutions came up. Following the contact from New Idria to the southeast there are the Sulphur Spring, Molina (Aurora) and San Carlos mines and prospects near the New Idria, and Del Mexico some 9 miles away. Coming northwest on the southern side of the serpentine belt are the Don Miguel, Picacho, Alpine and Monterey group of prospects. Of these only the San Carlos has been thoroughly prospected and exploited. It belongs to the New Idria company and the ore is taken to Idria over an aerial tramway. The contact area around the serpentine is evidently mineralized. It represents 40 miles of potential ore-bearing ground provided that structures favoring a concentration of the primary mineralization can be found.

Oat Hill

The Oat Hill quicksilver mine, about 9 miles southeast of Middletown, Napa County, Calif., is the third most productive mine in the state.

The rocks of the region around Oat Hill belong to the Franciscan formation which here is made up of flat-bedded sandstones intruded by peridotites at some distance from the mine itself and which have since been altered to serpentine. The Franciscan series is of great extent in this region and includes practically all the quicksilver mines in this

part of California, from the Cloverdale on the west to the Knoxville on the east, the mines around Clear Lake and others south to Napa. A description of the geology of the region is given by Osmont[30] who correlates Becker's descriptions with the later observations. Davis[31] reviews the interesting question regarding the age of the Franciscan and Knoxville series. Since Becker's time little or no geologic mapping has been done in this region, perhaps because of the fact that not even topographic maps have so far been made of any part of this entire quicksilver-mining region

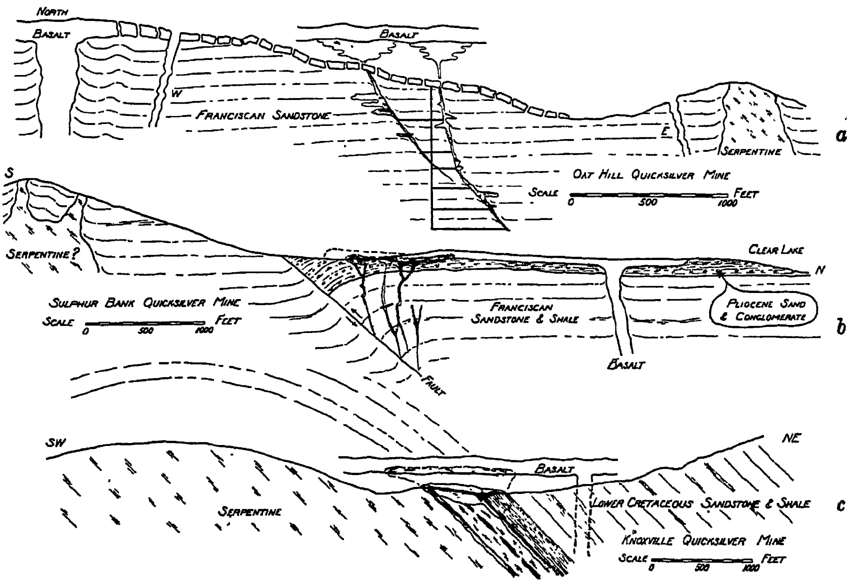

FIG. 4.—ORE OCCURRENCE AT OAT HILL, SULPHUR BANK AND KNOXVILLE MINES.

North of the site of the Oat Hill mine an extrusion of basalt reached the surface and spread out over it, covering the surface over the mine with a large but relatively thin sheet. The sandstone beds were fractured during this extrusion, the fissures having an approximately radial direction from the basalt core. Contortion of the sandstone beds can be noted in the underground workings as the basalt core is approached. At the present time the sheet of basalt which once covered the surface over the mine workings has been broken into fragments which are scattered over the entire slope of the hill.

The orebodies mined were the fissure fillings of some eight major fractures in the sandstone, together with impregnated zones in the more porous strata of the sandstone.

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[30] V. C. Osmont: A Geological Section of the Coast Ranges North of the Bay of San Francisco. Univ. Calif. Dept. Geol. *Bull.* (1904–06) **4**, No. 3.

[31] E. F. Davis: The Franciscan Sandstone: Univ. Calif. Dept. Geol. *Bull.* (1918) **11**, No. 1.

Fig. 4*a* illustrates the ore occurrence at Oat Hill. Many of the fissures were worked out to the surface and it seems probable that formerly they extended upward above the present surface and were capped by the basalt flow. This supposition is indicated on the section and much larger orebodies than those mined were probably lost by erosion and deposited as placers in the valley below.

In this connection it is interesting to note that placer mining for cinnabar has been carried on in a small way below the Oat Hill mine for many years. This placer cinnabar can hardly come from the mine dumps, as no placers of similar extent are found below the dumps of other large quicksilver mines.

The east-west section shown was chosen because it shows the main shaft and the deepest level of the mine. Most of the other fissures were mined by drift tunnels, the steep hillside favoring this form of mining. The ore-filled fissures run north toward the basalt core and the reader is asked to visualize the core shown on the left of the illustration as lying behind the section in line with the shaft. It may be that the basalt core is actually in the form of a long dike running east and west and that the major fractures are roughly perpendicular to its strike and are spaced at fairly even distances apart over a total length of about one mile. Forstner's[32] sketch of the fissures suggests this and also shows a number of cross fractures.

The history of the ore occurrence at Oat Hill was probably as follows. The basalt extrusion fractured the Franciscan sandstones, the fractures being approximately at right angles to the dike of rising basalt. On reaching the surface the flow spread out, covering the surface and sealing over the fractures in the Franciscan rock. As the final phase of the extrusion hot alkaline magmatic solutions carrying quicksilver in solution rose through the fractured zone along the basalt-sandstone contact. As they reached and entered the transverse fissures they became cooler and lost pressure on account of the porosity of the rock. Traveling upward in the fissures they penetrated and impregnated the more porous portions of the sandstone wherever clay fracture gouges did not hinder them. The fracture gouges in this mine, by the way, are as often on the footwall as on the hanging wall. The fracturing was accompanied by some vertical displacement as the strata are tilted up on the hanging wall and down on the footwall in some of the fissures. The ascending solutions were finally stopped by the basalt capping and concentration of the primary mineralization continued as long as new mineral was carried in by the solutions.

The ore does not extend back to the basalt core, as the residual heat of the latter probably prevented precipitation from the mineral-bearing solutions.

[32] W. Forstner: *Op. cit.*, 89.

In general the strike of the fissures was toward the basalt intrusion; the dip was east, the grade of the ore depended on the porosity of the receptacle rock and orebodies formed by impregnating the wall rock on the side opposite the one carrying the fault gouge.

Other mines in the vicinity have a similar history and here as at New Almaden and New Idria a promising territory remains to be explored.

Knoxville

Ranking next below the Oat Hill in point of production is the Knoxville (formerly Redington, then Boston) mine at Knoxville in the northeast corner of Napa County, California. The orebodies lie near a contact between serpentine and shales and sandstones. The serpentine here, as at New Idria, is a large intrusion, probably a laccolith. Unlike New Idria the Franciscan rocks at Knoxville into which the serpentine was intrusive had been removed by erosion and here Lower Cretaceous shales and sandstones rest unconformably on the serpentine. A doming of the rocks overlying the serpentine is indicated here also. The strike of the contact at the mine is northwest and southeast. The shales and sandstones dip northeast and may be the northeast end of an anticline formed by the peridotite intrusion. This condition is indicated in Fig. 4*c*, which illustrates the ore occurrence at this mine. Directly northwest of the mine covering the contact and at a slightly greater elevation than the present surface at the mine is a sheet of olivine basalt. The geology of the region was mapped by Becker[33] and a section of the mineralized zone is given by Forstner.[34] From these sources and personal observations Fig. 4*c* was drawn, on which the basalt flow is indicated. The stock or core of this extrusion was found to be a dike and to lie near the contact but probably on the shale side of it as indicated in the section.

The lowest member of the Cretaceous rocks is a bed of shale 100 to 200 ft. thick. Both it and the serpentine in contact with it are fractured by movement and a heavy attrition gouge lies between them. To the southwest—that is, under the contact fault—there is a second parallel fracture or brecciated zone, which also is marked by an attrition gouge, though not as heavy a one as the upper. A third fracture lies midway between these two but no gouge was formed, the fissuring being apparent by an irregular brecciated zone some 100 ft. wide.

Here again, as at Oat Hill, a later eruptive came up along the plane of weakness followed by the peridotite intrusion.

The release of hot alkaline magmatic solutions was probably the final phase of this second period of magmatic activity. Since the dike or stock of the basalt was in, close to, or perhaps cut through the contact, the mineralizing solutions in their upward course penetrated and rose

[33] G. F. Becker: *Op. cit.*, atlas sheet V.

[34] W. Forstner: *Op. cit.*, 78.

through the brecciated zones along the contact under the attrition gouges and through the brecciated zone between them until arrest by the basalt sheet above. Here again is a trap structure compelling a concentration of the primary mineralization.

If, as evidence indicates, the basalt dike cut through the sandstones and shales and if the solutions followed the same fractures we would expect some mineralization in the sandstone-shale series. Actually cinnabar is widely disseminated through the sandstone northeast of the contact. No orebodies were formed here, however, because the structure necessary to trap the solutions was absent.

The probable original form of the orebody is indicated in the drawing above the profile of the present surface. Much of this may have been lost by erosion before the remainder was accidentally found by cutting through it during the construction of a county road.

The upper part of the mine was a jumbled mass of fractured rock and had apparently been exposed to surface agencies for a considerable period of time. Pyrite was deposited with cinnabar by the magmatic solutions. Acid surface waters redissolved some of the cinnabar, and sulfurous gases, probably H_2S, reprecipitated the dissolved cinnabar as the black sulfide or metacinnabar. Marcasite, which is formed from pyrite by the same agencies, is found in the upper part of the ore zone also. The action producing the secondary quicksilver mineral was probably localized on the surface and in the cracks of the cinnabar and gave the ore the black appearance that has given rise to the descriptions of this orebody as an immense deposit of metacinnabar. It is barely possible but not at all probable that all of the cinnabar was altered to metacinnabar.

Round about Knoxville, as in the vicinity of the Oat Hill mine, there are numerous mineral springs of a probably solfataric origin. Solfataric gases were encountered in the mine and at places sulfur crystals were formed on the timbers. Erosion is also indicated at Knoxville as cinnabar can be panned in the vicinity as well as fine gold which was derived from pyrite dissolved by the acid waters. In view of recent experiments on the production of gold from quicksilver and repeated statements in the press that gold and quicksilver are never associated in nature, it may be well to mention that the pyrite accompanying cinnabar in the quicksilver mines of California nearly always carries gold, that many quicksilver ores are known that carry up to $2 in gold value and that at least one mine recovered both gold and quicksilver from the ore.

There are other quicksilver mines and prospects in the vicinity, that is around the edge of the basalt flow, and a similar history is indicated for them. One of them has the ore under a basalt capping in "mudrock," which is probably the lower member of the Cretaceous rocks.

Altogether some 1000 acres are covered by the basalt flow. The known quicksilver deposits were found along the edge of the flow, probably

because they were exposed here by erosion. It is probable that other orebodies exist under the basalt capping and a critical study might point to areas that it would pay to prospect by churn drilling or shaft sinking. If existing geophysical methods of prospecting are applicable to cinnabar deposits, or at least to the location of fissures or dikes, this method might be used to advantage in locating the areas to be drilled. The possibility of large high-grade orebodies lying relatively close to the surface is certainly a favorable condition to prospecting by either method; and an incentive to risk such a venture.

Sulphur Bank

Next in rank as measured by total production in comparison to that of mines previously described is the Sulphur Bank mine on the eastern shore of Clear Lake in Lake County, California. It is about 10 miles north of the town of Lower Lake.

Like so many of the other important quicksilver mines this one was discovered by accident, the mine having been worked originally for sulfur, a deposit of which covered the cinnabar orebody. The country rock in this vicinity is the same series of Franciscan sandstone and shales as at the Oat Hill some 25 miles south of it. Becker notes the occurrence of serpentine on the ridge south of the mine but it was not possible to confirm this observation.

On the Franciscan at the site of the mine is a series of fresh-water sands and conglomerates late Pliocene in age, called the Cache Lake beds. At the mine these strata are an erosional remnant and are covered in part by a basalt flow. A generalized section of Sulphur Bank is given in Fig. 4*b*, which was drawn from Becker's atlas sheet IV and Forstner's[35] description. The fault shown on the section runs roughly east and west and its easterly end outside the area covered by the basalt is marked by areas bare of vegetation and the emanation of sulfurous gases.

The basalt extrusion probably followed the plane of weakness marked by this fault until near the surface, when it broke through as indicated. It may have followed the fault plane to the surface or its source may have been farther away, depending on whether the various basalt sheets in the vicinity had a single source or were flows from separate dikes.

Solfataric action here seems more recent than at Knoxville. Hot water and gases issue at various points in great quantity. Rumbling and roaring sounds are quite audible at a number of the vents.

The history of these deposits is similar to that of those at Oat Hill and Knoxville, but at Sulphur Bank the process can be seen in action. The mineralizing solutions rose along the fault fissure and no doubt along the stock of the basalt flow also wherever that may be located. Movement along the fault plane at the time of the basalt extrusion seems probable.

[35] W. Forstner: *Op. cit.*, 64–65.

Near the surface the hanging-wall block above the fault developed rifts through which the solutions sought escape only to be trapped under the basalt flow. The basalt on cooling developed shrinkage cracks in the manner of the well known pillow basalts. The solutions deposited cinnabar in these openings as well as in the brecciated rock below. The top of the original bank was white silica, the decomposition product of the basalt. This was produced on top where oxygen of the atmosphere combined with the sulfur vapors to form acid. Lower down where the atmospheric oxygen did not penetrate, sulfur was deposited. Below this zone is the cinnabar deposit.

Sulfur and also cinnabar are still being deposited from a few active solfataras that reach the surface in the form of hot springs bubbling with gases.

Toward the south the basalt flow was either removed by erosion or perhaps it never reached to the fault. Here, along the edge of the basalt flow, the six working shafts of the property were sunk. Most of the production of Sulphur Bank came from the surface ores. The ore in the underground workings was found in brecciated zones through which hot waters and gases were still rising.

Several other basalt flows in the vicinity are worth prospecting. Location of the dike or stock of the basalt might lead to the finding of radial fissures similar to those at Oat Hill. The area of basalt flows near Sulphur Bank is at least equal to that near Knoxville.

The three mines just described form an interesting trio. At Oat Hill not only the cap rock but also the probable original orebodies have been removed by erosion, leaving only the feeder fissures and impregnations from them into the porous sandstones. These feeder fissures were found by prospectors following their pannings up the hill slope from James Creek below. At Knoxville the cap rock had been removed but part of the orebody formed under it was still in place when it was accidentally discovered during road-building operations. At Sulphur Bank the cap rock was still in place and ore deposition was active when it was accidentally found while mining the overlying sulfur. Sulphur Bank did not outcrop, Oat Hill and Knoxville were exposed by erosion. Large areas of basalt flows remain which may be capping many other equally rich deposits of quicksilver ore. Here are areas in which geophysical methods of prospecting could be put to a fair test as large deposits at shallow depth are possible.

Cloverdale

The Cloverdale quicksilver mine, some 13 miles east of Cloverdale, Sonoma County, Calif., on Big Sulphur Creek, is a typical example of the "exceeding irregularity," as noted by observers, of quicksilver deposits.

This orebody lies under a heavy attrition gouge on a sandstone-chert contact of Franciscan rocks. The general strike of this contact is northwest-southeast and it dips some 35° to the northeast. Locally almost any dip and strike may be measured. Cross faults and fractures are numerous.

The ore occurrence at this mine is illustrated in Fig. 5, which is fairly accurate though prepared from two maps that were not in close agreement. The outcrop and the underground contours of the fault gouge are shown.

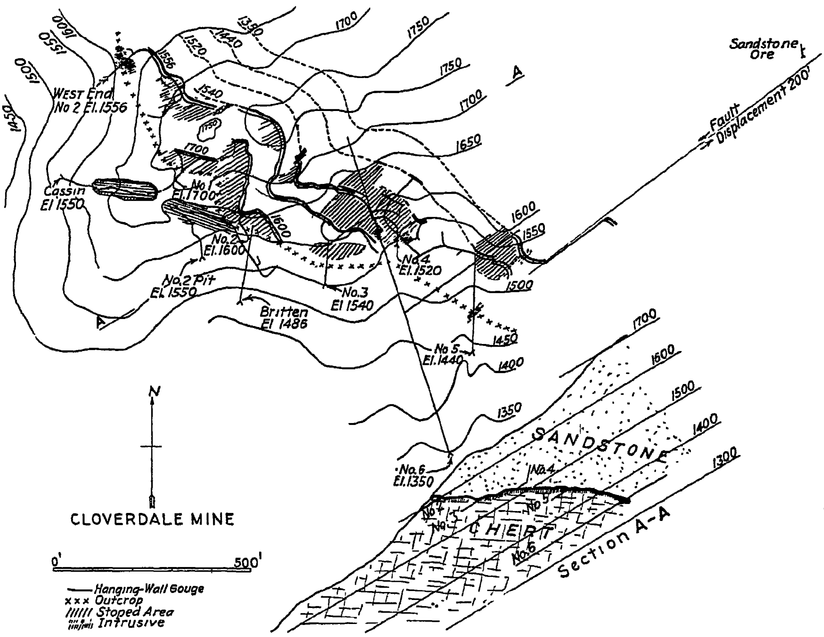

FIG. 5.—ORE OCCURRENCE AT CLOVERDALE MINE.

The ore solutions seem to have come up along dikes of intrusive diorite (?) rock the altered remains of which are found on many levels and locations through the mine. Seven exposures of this intrusive are shown on the map. The ore is found near but not in contact with the intrusive. In general the course of feed fissures was perpendicular to the general strike of the fault gouge. The chert underlying this gouge is the typical radiolarian chert of the Franciscan series first described as such by Ransome.[36] This chert occurs in beds from ½ to 3 or 4 in. thick, separated by seams of shale. The chert in the beds is broken into blocks by cross fractures in two directions so that the aggregate interstitial

[36] F. L. Ransome: The Geology of Angel Island. Univ. Calif. Dept. Geol. *Bull.* (1894) 1, No. 7, 198.

space is great. The mineralizing solutions coming up along the cooling dikes were deflected by the clay gouge and caused to rise through the chert beds under it. The deposition was heaviest just under the gouge and the ore deposit grades off from this hanging wall to a commercial footwall. The movement along the contact seems to have been the first disturbance. After the contact gouge was formed further movement apparently caused distortion, resulting in the crumpled appearance shown on the map. Further premineral fracturing is indicated by many ore-filled fractures across the strike of the ore and a fault of some 200-ft. displacement on No. 6 level. The Cassin workings are now a large glory hole in which the stratum of chert has been thrown into a nearly vertical position; No. 2 Pit and Britten workings are related to this fault fracture, as here the ore was stoped for a length of 120 ft. to a depth of 90 ft. and some four sets wide. These workings, as can be seen from the map, are outside the main ore occurrence under the gouge although both Cassin and Britten workings were probably originally sealed by an extension of the gouge which has since been removed by erosion.

A casual visitor at the mine is soon confused by the continual change in direction of drifts, dips, strikes and ore value underground and comes out firmly convinced that the whole thing is a hopeless jumble. A study of the map shows that the ore occurrence is fairly regular in that it is confined, with the exception of Cassin and Britten, to the layer of chert under the contact gouge. This, while irregular in direction, is persistent and easily followed. Here also the orebodies were formed because the requisite conditions for a concentration of the primary mineralization obtained.

Quicksilver Rock

In none of the mines thus far described has the ore been associated with the famed "quicksilver rock" mentioned in the writings on California quicksilver mines. Three mines in which it is found are the Great Western, Helen and Great Eastern. They are all in the Franciscan formation. The Great Western in Lake County has been described by Becker[37], who also gives a geologic map of the region. The mine lies on the slope northwest of Mount St. Helena near the edge of lava flows from this old volcano. The ore occurs near a contact of chert and serpentine. Movement along the contact brecciated the serpentine near the contact zone. This zone was opalized by a flow of silica-carrying solutions. At a later date alkaline solutions carrying cinnabar rose along the same plane of weakness through the shattered chert, the opalized rock and the serpentine being impervious. The ascent of the solutions was limited by the lava flow then covering the surface. Becker states that the ore was for the most part enclosed on three sides by serpentine.

[37] G. F. Becker: *Op. cit.*, 358.

By this he means that one wall of the orebody was serpentine and that the ends of the individual oreshoots on the strike as he mapped them were limited by the bulging serpentine undulations on the contact. The strike of the ore zone was N. 65° W. and the dip changed from southwest at the top to northeast below. One wall was serpentine and the other graded off. The grade of the ore was highest where the chert was most shattered. Evidence of silicification preceding or accompanying ore deposition can be noticed at other quicksilver deposits also and has been discussed extensively by Becker.[38] The Helen mine lies on the slope of Pine Mountain 4 miles northwest of the Great Western mine. From Pine Mountain a lava flow extends to the ground of the Helen mine. The strike of the ore zone in this mine is northwest-southeast and the dip is 30° to 35° to the southwest. The hanging-wall rock is sandstone, the footwall serpentine. Movement along the contact formed a heavy gouge. Silica-bearing solutions rose under this gouge and through the brecciated serpentine, depositing opaline silica or quicksilver rock. Three orebodies have been worked, two of them lying under this gouge and a third one coming up vertically through the serpentine. On the strike between the first two oreshoots a greatly altered intrusive is exposed in the workings. The oreshoots are some distance from it, as was the case in the other mines described, showing that cooling was probably the main factor influencing deposition. An eruptive resembling basalt is exposed on the surface and may be the same one as that exposed underground.

The Great Eastern mine is near Guerneville in Sonoma County, California. This mine is on a sandstone serpentine contact also. The strike is N. 70° W. and the dip some 50° to 60° N. The serpentine here was originally a dike of peridotite which came up along a plane of weakness in the sandstone formation. Later the silicification by ascending solutions along the same course formed the dark opaline rock mass along the contact with the serpentine. Renewed movement brecciated the opalized rock mass and opened channels for the ascent of the quicksilver-bearing solutions, and this seems to have been accompanied by further silicification, as large untimbered stopes can be left in the silicified sandstone orebodies in the footwall. The ore-bearing solutions seem to have followed dike rocks here also. The decomposition products of this intrusive here, as in the other mines described, is a soft light-colored mushy mass. In this mine the trap for the genetic solutions was an irregular brecciated pipelike zone developed in the highly silicified brittle rocks.

The "quicksilver rock" probably achieved notoriety by being harder than its neighboring rocks. It forms bold croppings that can be pointed to with pride. Actually the ore deposition always came later and became associated with the quicksilver rock only when and where the magmatic

[38] G. F. Becker: *Op. cit.*, 392 and Chap. III.

ore solutions rose along the same plane of weakness through which the siliceous ones had preceded them. Many large persistent ledges of opalized rock in California have so far failed to show even a trace of cinnabar despite assiduous digging. Promoters who are about to pull a million dollar profit rabbit out of the hat by identifying an opalized ledge as quicksilver rock should be obliged to show the color of their cinnabar first. Quicksilver mines may contain "quicksilver rock" but the converse is not necessarily true.

Oceanic Mine

The Oceanic is the largest producer in still another quicksilver-mining district of California. It lies 5 miles east of Cambria in San Luis Obispo County. The geology of the region has been described by Fairbanks.[39] The general region is composed of Franciscan rocks on which rest remnants of Miocene strata, or, as Ransome[40] thinks, are faulted against it.

The geology has never been satisfactorily worked out, though the operators would undoubtedly benefit by doing this. Published descriptions[41–43] reflect this uncertainty by exhibiting numerous discrepancies. Much of the uncertainty arises from the fact that Becker supposed the serpentines to be altered sedimentary rocks and he speaks of partly serpentinized rock; that is, of rocks that according to his idea were in process of alteration to serpentine. This has led to great confusion, as older writers classified many rocks as serpentine that are nothing of the sort.

The serpentine of the Franciscan is derived from peridotites and these peridotites were intrusive into the Franciscan only. The Franciscan is characterized by lack of fossils while many fossils are found in the mudrock ore of the Oceanic mine. These fossils are believed to be Miocene in age.

Fig. 6*a* shows a longitudinal section after Bradley[44] and 6*b* shows a cross-section. The ore occurrence strikes northwest and southeast. The dip near the surface is southwest and farther down it reverses and dips northeast. The ore zone varies from 20 to 60 ft. in width. The hanging wall is a thick gouge above and a mere slip in the mudrock below. Bradley also mentions serpentine as being the hanging wall. The foot-

[39] H. W. Fairbanks: Geology of Northern Ventura, Santa Barbara, San Luis Obispo, Monterey and San Benito Counties. Twelfth Report of the State Mineralogist, California State Mining Bureau (1893–94).

[40] F. L. Ransome: Quicksilver. U. S. Geol. Survey, *Min. Res. of U. S.*, 1917, Pt. 1, 388.

[41] C. A. Heberlein: Mining and Reduction of Quicksilver Ore at the Oceanic Mine, Cambria, Cal. *Trans.* A. I. M. E. (1915) **51**, 110.

[42] W. W. Bradley: *Op. cit.*, 142.

[43] W. Forstner: *Op. cit.*, 162.

[44] W. W. Bradley, *Op. cit.*, 142.

wall is barren sandstone or mudrock, a darker finer grained sandstone. In other words, the ore grades off from the heavy gouge on the northeast or hanging wall to a commercial footwall, depending on the price of quicksilver. Forstner[45] mapped serpentine and igneous, which he describes as rhyolite north of the mine. Bradley mentions a diorite gabbro on the southwest of the mine and more or less parallel to the orebody. Fairbanks[46] describes a number of analcite diabase dikes paralleling the orebody southwest of the workings and remarks on the high percentage of alkalies they contain. He presents evidence that they have a common source. A composite of these observations is shown in Fig. 6*b*. Carson[47]

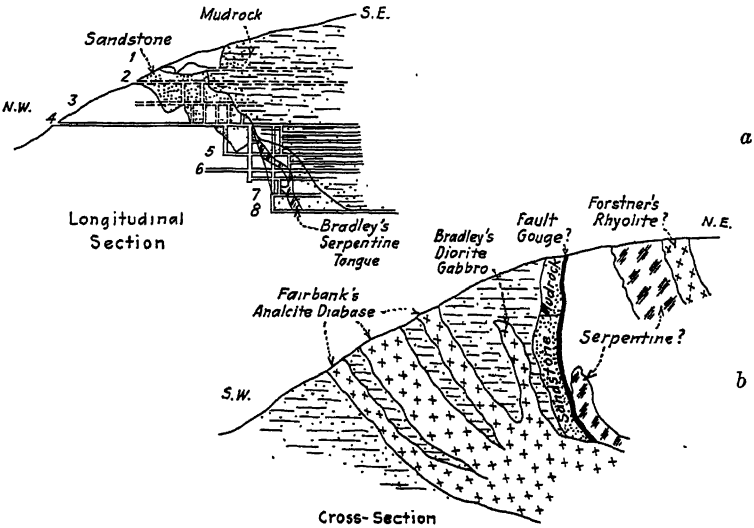

FIG. 6.—SECTIONS OF OCEANIC MINE.

states that on No. 8 level all sedimentaries disappeared. A drift was run east following the contact of diorite and what was locally termed serpentine. The diorite was on the south and the serpentine on the north. No mudrock or sandstone was found but on the lower levels the mudrock orebody rakes to the east at a rather flat angle and may have been farther to the east on the bottom level. If the occurrence of serpentine as drawn in Fig. 6*b* is correct the strata northeast of the fault gouge would be Franciscan with Miocene strata on the southwest. Bradley's serpentine tongue, however, would have to be in the hanging wall or else not serpentine. The latter is the more probable.

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[45] W. Forstner: *Op. cit.*, map opposite 148.

[46] H. W. Fairbanks: On Analcite Diabase from San Luis Obispo County, California. Univ. Calif. Dept. Geol. *Bull.* (1893–96) **1**, No. 9.

[47] E. Carson (formerly manager of the Oceanic): Private communication to the author.

Be that as it may, the manner of formation of the orebody is clear. The mineral-bearing solutions probably followed the diabase intrusion. They rose through the coarse-grained sandstone until deflected by the mudrock. The mudrock being only relatively impervious was also impregnated, though to a lesser extent. Here one has a splendid example of the influence of the porosity of the receptacle rock on the grade of the ore. The coarse-grained sandstone ore averaged better than 1 per cent., while the mudrock will average about 0.25 per cent. Many shark-tooth fossils are found in the mudrock and shells that are difficult of identification. The latter are changed to cinnabar and generally contain a bit of native quicksilver. The analcite dikes can be traced for at least 12 miles and there are several prospects along their course. Boulders of diorite (diabase?) are found on the slopes of Pine Mountain 11 miles northwest of the Oceanic, which suggest that the later rhyolite flows of this locality were ejected along planes of weakness that had in earlier times served the diabase dikes as vents.

The San Luis Obispo district has thus far produced some 43,000 flasks of quicksilver. There are widespread indications of cinnabar and a geological study of the region might indicate several localities worth a closer study and more intensive prospecting than has been accorded them.

Quicksilver Deposits in Texas

Cinnabar was first found in Texas near the present town of Terlingua, Brewster County, in 1894. Since then further discoveries have been made over an area extending some 50 miles northwest and southeast and 40 miles in northeast and southwest direction. The rocks of the region range through the Upper and Lower Cretaceous ages. The district lies at the intersection of one of the cordilleran zones of faulting and a minor east and west zone of faulting.[48–49]

The presence of deep-seated active magmas in this region is indicated by intrusive rocks and numerous hot springs. That the ascending mineralizing solutions in this district have been arrested in their upward course by impervious formations and that the orebodies were formed in the voids of the rock under these impervious caps has been recognized and mentioned by a number of geologists.[50–52]

[48] R. T. Hill: The Geographic and Geologic Features, and their Relation to the Mineral Products of Mexico. *Trans.* A. I. M. E. (1902) **32,** 172.

[49] J. A. Udden: Structural Relations of Quicksilver Deposits. *Min. World* (May, 13, 1911) 975.

[50] B. F. Hill: The Terlingua Quicksilver Deposits. Univ. Texas Mineral Survey *Bull.* 4 (October, 1902).

[51] H. W. Turner: The Terlingua Quicksilver Deposits. *Econ. Geol.* (1905–06) **1,** 255.

[52] J. A. Udden: *Op. cit.* and The Anticlinal Theory as applied to Some Quicksilver Deposits. Univ. Texas *Bull.* 1822 (Apr. 15, 1918).

The formations related to the ore deposits to be described are the Taylor marl, Austin chalk, Eagle Ford, Buda, Del Rio and Edwards in descending order. The last three are Lower Cretaceous, the others Upper Cretaceous in age.

Mariposa

The first prominent producer of the Texan district was the Marfa and Mariposa mine. A section illustrating the ore occurrence is given on Fig. 7*a*. The structure suggests that the strata have been raised and folded by the injection of an igneous sill or laccolith. Fracturing of the overlying strata accompanied the uplift. In the coarse-textured massive Edwards limestone the fracture fissures are large and open while in the overlying Del Rio clay they practically sealed themselves. The mineral-bearing solutions rising from the magma ascended through the fractured Edwards until dammed by the Del Rio, which everywhere formed an impervious cap rock. The solutions trapped under this cap deposited their load of mineral matter in concentrated form. Erosion later removed the upper sediments except where a few remnants of the thick Buda limestone protected the soft Del Rio clay from erosion at what is now known as California Hill and other points.

A large part of the production of this mine came from what might be called the detrital deposits left on, and in irregularities of the surface of the Edwards limestone, where the Del Rio clay, the original cap rock, had already been removed by erosion. Later underground mining developed orebodies under and against the Del Rio clay and as far down the feeder fissures as the ore persisted.

Many rare secondary minerals of quicksilver were found in the shallow surface pits where conditions favoring their formation obtained. Broderick[53] has shown the part that calcite plays in their formation. The mode of occurrence of the secondary minerals seems to emphasize the fact that the orebodies as originally formed were primary mineral and that exceptional conditions are necessary to form anything except cinnabar even in the relatively small quantities in which the secondary minerals were obtained here.

One interesting orebody was found on section 38, which apparently violates all the precepts of the theory advanced in this bulletin. This is an open fissure in the Edwards limestones that has been filled from above by erosional detritus. The material filling the fissure consists of rounded boulders and pebbles of cinnabar, mixtures of fine cinnabar and clay, barren streaks of clay, occasional rounded fragments of stalactites, and worn fragments of what appears to be Buda limestone. Sharp angular fragments of Edwards limestone have been found that had apparently spalled off the walls. Fossil bones are occasionally found.

[53] T. M. Broderick: *Op. cit.*, 651.

A feature particularly noticeable on the upper levels were pot holes that had been worn into the sides of the walls. All the loose material in the

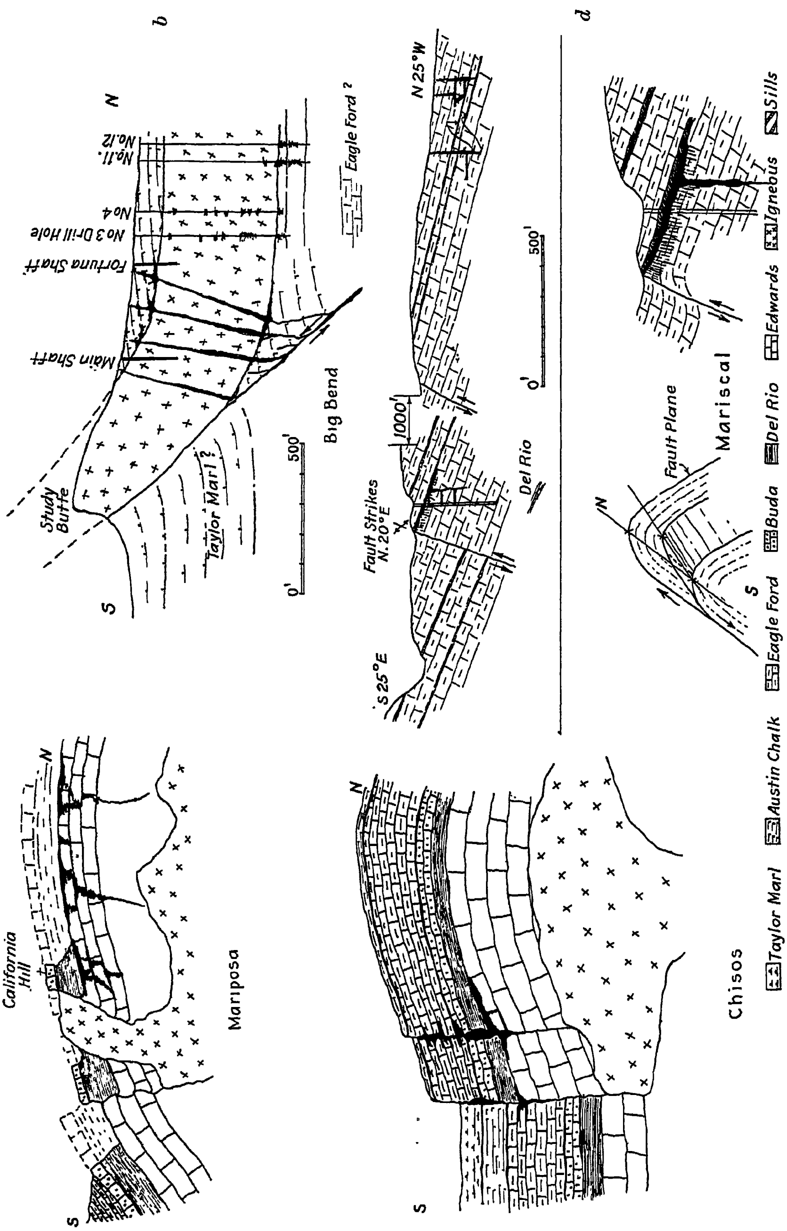

FIG. 7.—QUICKSILVER DEPOSITS IN TEXAS.

fissure is taken out for ore. Apparently this fissure has remained open for a long period of time. Buda and Del Rio both eroded into it. This

accounts for the cinnabar boulders and the clay-cinnabar mixtures. Occasional cloudbursts may have dropped a boulder into the fissure and the rush of water moving the boulder accounts for the pot holes. Then the rush of water along the fissure died down, its carrying power decreased and the barren streaks of clay resulted.

Big Bend and Dallas

The Big Bend and Dallas mines at Study Butte lie in the highest rocks, geologically speaking, in which quicksilver is found in this district. A section of the mine is shown in Fig. 7*b*. The mine is under Study Butte, a prominence formed by a hard andesite that intruded the soft Upper Cretaceous sediments.

This andesite intrusion, probably a laccolite or a large sill some 400 ft. thick, spread horizontally through the Austin chalk formation. At Study Butte there seems to have been a thrust fault, the fault plane dipping some 45° north, which threw the Taylor marl against the Austin chalk. The andesite intrusion pushing south reached this plane of structural weakness and followed it up as indicated. Because of the greater hardness of the intrusive it now forms a prominent butte with a steep cliff facing south. As the great body of molten andesite cooled after intruding, shrinkage cracks formed in the neck as indicated.

The ore solutions probably came up along the fault fissure from their magmatic source farther down. Deposition took place in the shrinkage cracks and brecciated areas in the andesite and in such open spaces of the underlying and overlying rock as they reached. The Austin chalk formation contains close-textured shales and except for brecciated areas near the contact formed an effective seal. It seems possible that the Eagle Ford formation may be found at no great depth below the andesite on the north side of the fault. This would be an area well worth prospecting, as these harder and more open-textured rocks, overlain as they are by close-textured shales, would form a splendid receptacle rock for orebodies. Ore on the lower andesite contact north of the mine workings is indicated by diamond-drill holes, as shown in Fig. 7*b*. The fissures in the andesite in which the ore is found dip steeply and are practically parallel with a northwest and southeast strike.

Chisos Mine

The Chisos mine lies on the southern leg of an anticline. The anticline may have been formed by a laccolite intrusion, as indicated in Fig. 7*c*. Along the southern edge of the anticline a long curving rift block, about 9 miles long and 1 to 2 miles wide, sheared off. It shows a vertical displacement of 1000 ft. with a lagging block which has a displacement of 40 to 60 ft. with the main block of the anticline. At

the Chisos mine cross fractures have developed, caused by a nosing down of the anticline toward the east at this point.

The ore solutions ascended through the fault fissures and were concentrated in the broken Edwards limestone under the impervious Del Rio clay, Fig. 7*c*. Fortunately, in the case of this mine the Del Rio clay, because of the fault fracture, was not a perfect seal and allowed some of the ore solution to rise through it into the overlying rocks. Had this not happened the orebodies would have lacked an outcrop and might never have been found. The fault movement between the lagging block and the main block of the anticline had a horizontal as well as a vertical component. By the juxtaposition of oppositely curved wall sections a vertical opening was formed through the Del Rio, into which fell sufficient coarse material from the Buda limestone to maintain a channel for the ore solutions. Ransome[54] suggests that perhaps a section of Buda limestone was faulted down through the Del Rio as the consequence of the collapse of a solution chamber in the Edwards.

Whatever the cause, a channel was formed and through it the ore solution rose along the fault fissure into the upper shales and deposited their mineral load in the fractured areas near the main fault fissures and cross fissures. To date the mine has been explored and developed through 16 large and small shafts. The more important shafts lie along the fault zone and ore has been found over a length of nearly 3000 ft. and to a depth of 900 ft. The Austin chalk shales were probably the seal for the rising solutions. When the upper levels were satisfied ore deposition took place in the channel through the Del Rio. This opening was a roughly circular chamber some 75 to 100 ft. across and 150 ft. in vertical extent. Being filled with a breccia of Buda limestone and clay-shale fragments, the interstitial space was great and this rock mass furnished an ideal receptacle for ore concentration and deposition. The orebody so formed was the richest one of the mine. Of late years the Del Rio-Edwards contact is being explored. As is to be expected, the ore here is rich but a large flow of water must be handled at this horizon and this retards operations.

Mariscal

This mine lies some 40 miles southeast of Terlingua and the other Texas quicksilver mines, in the toe of the Big Bend of the Rio Grande. Mariscal Mountain, from which the mine takes its name, is a long anticlinal fold across which the Rio Grande has cut a steep canyon. This anticlinal fold, highest in Mexico, slopes downward toward the northwest. This mine is some 8 miles north of the river, where the anticline pitches

[54] F. L. Ransome: Quicksilver. *Min. Res. of U. S.* 1917, Pt. 1, 422.

down under the general level of the country. At the mine the anticline noses down rather sharply with accompanying transverse faults and fissures. These were the feeder fissures for the orebodies. To date the mine is entirely within the Eagle Ford formation, though on going southeast on the crest of the mountain the Buda limestone and below it a few feet of marly material representing the Del Rio are exposed. Below this and extending to the river is the Edwards limestone. At the mine the Eagle Ford formation was intruded by sills of igneous rock which lie in the stratification. One of these formed the impervious cap rock needed to concentrate the mineralizing solutions that formed the main orebody. Other sills are present which cut the stratification.

The main orebody lies under its cap intrusive near a fault, as shown in Fig. 7*d*.

The trend of the ridge at the mine is N. 25° W. Looking north from the hill south of the mine the displacement of the fault which strikes N. 20° E. can be seen by observing the top of the anticline south and north of the fault. The displacement was apparently as indicated in the sketch of Fig. 7*d*.

Two thousand feet north of the main workings are a number of transverse fissures explored by four small shafts, a drill hole and some drifts and crosscuts. Some ore is found in these fissures and cross fissures but there is not exposed at this locality any structure that would be favorable to a concentration of the cinnabar into an orebody. Two small sills of igneous rock were cut by one of these shafts. A concentration of pyrite in a thin, dark, very hard dense rock was found under the lower one, but no cinnabar. Stratigraphically these fissures are higher than the main orebody.

The mine has had two periods of production, the first with retorts, the second with a furnace. Exploration so far has been confined to following known orebodies. The Edwards limestone under the Del Rio horizon remains to be explored, as does the fault south of the main shaft and the igneous sill under which the ore is found, on the south side of the fault. The north end of the property is also deserving of exploration in depth.

Both north and south of the mine traces of cinnabar have been found in place at various prospect holes, and placer cinnabar has been picked up on the slopes of Mariscal Mountain. Hot springs in the river east of the mine give evidence of comparatively recent volcanic activity. There are in the Texas quicksilver district many prospects that would seem to warrant a little digging but prospecting is not active.

At various places in the district deposits of volcanic ash are found. Fossils of saurians and fossilized trees are abundant. Clear double-refracting calcite—Iceland spar—is found in many veins throughout the district, and with cheap Mexican labor available it might be possible to develop a national source of this useful mineral.

Quicksilver Deposits in Nevada

Cinnabar was known to occur at many places in Nevada before any recorded production of quicksilver was attained. Becker[55] studied the cinnabar occurrence at Steamboat Springs and Ransome[56] describes several others. The topographic map in Ransome's report embraces six cinnabar-bearing localities. One of these is at the head of Black Rock Desert just off the map in the 118° 45′ meridian line; three others are in the Humboldt Range at Eldorado Canyon (below Humboldt House) in American Canyon about 12 miles a little south of east from Oreana and the Antelope Springs district, which lies on the lowest contour jutting out south from Buffalo Peak 18 miles due east of Lovelock; the other two are at Gold Banks about at the *O* of the printed name, and the Ruby group of claims about 3 miles in a northwest direction from Boundary Peak some 32 miles south of Gold Banks.

The Antelope Springs district was the principal factor in Nevada's record production for 1928, when the state produced nearly 3000 flasks.

Two properties, the Nevada Quicksilver Mines, Inc., and the Pershing Quicksilver Co., were the active producers in this district in 1928.

Nevada Mine

The ore occurrence of the Nevada is illustrated in Fig. 8*a*. The oreshoot being worked lies under a shale on a limestone-shale contact, which strikes N. 30° W. with a dip a little less than 45° NE. There has been movement along this contact and two directions of fracturing developed, as shown on the sketch. Large fractures roughly perpendicular to the strike occur also and it was on an insignificant outcrop of one of these that the mine was discovered. At the time of this writing the orebody connected with this cross fracture has been stoped to the 250-ft. level. The shaft is 350 ft. deep and apparently another orebody has been found at this depth. The main oreshoot pitches north across the shaft. High-grade ore was found near the intersection of the contact and cross fissure as the ground was brecciated and afforded room for deposition. A fault displaces the contact some 20 ft. vertically between the 250 and 300-ft. levels. The limestone-shale contact outcrops for a long distance and cinnabar is found at a number of points along the outcrop. The relation of this deposit to igneous rocks is not yet clear, but there are basic igneous rocks in the vicinity and the structure of the formations indicates underlying igneous masses. It may be that a dike came up through the fault, displacing the contact between the 250 and 300-ft. levels, and that this was a feeder fissure for the overlying orebody. Cross fractures and

[55] G. F. Becker: *Op. cit.*

[56] F. L. Ransome: Notes on Some Mining Districts in Humboldt County, Nevada. U. S. Geol. Survey *Bull.* 414 (1909).

fractured areas in the limestone under the shale capping should be favorable places for ore deposition.

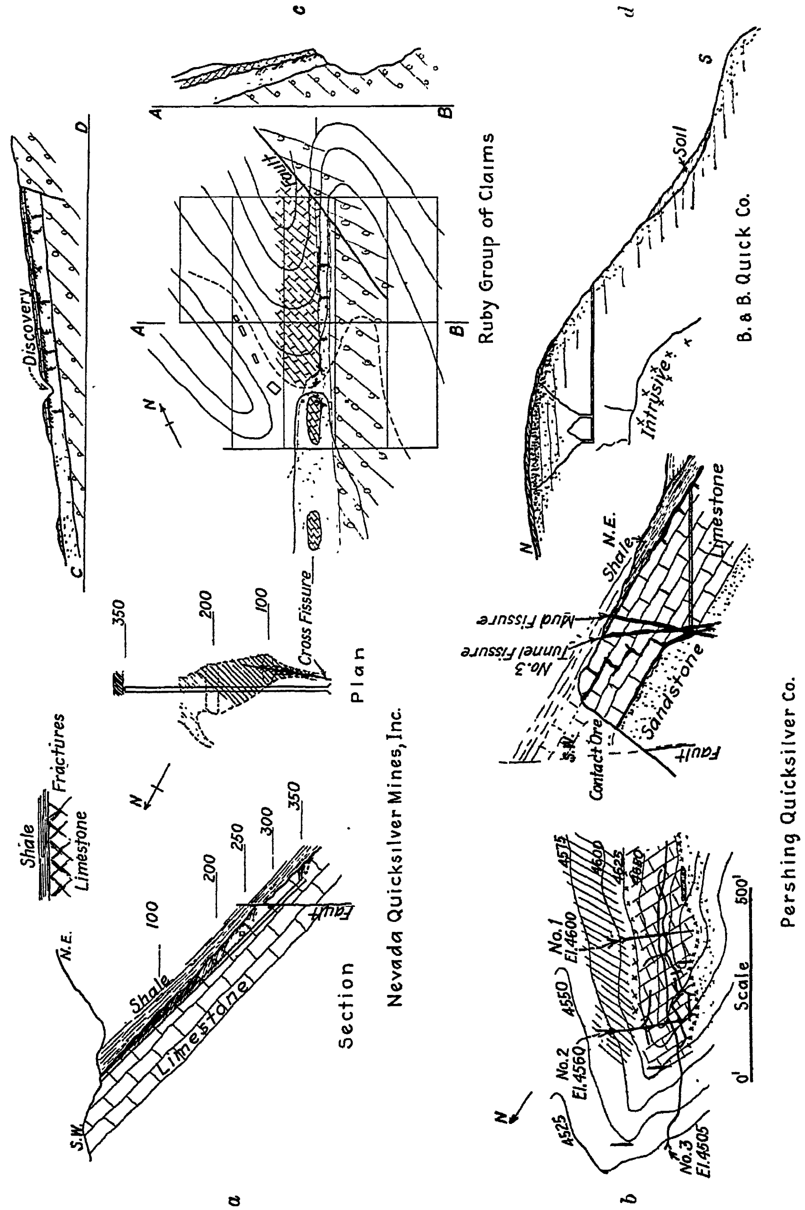

FIG. 8.—QUICKSILVER DEPOSITS IN NEVADA.

Pershing Mine

At this mine shale, limestone and sandstone are the country rocks. Fig. 8*b* shows a plan and a generalized section of the mine area. The

strike of the ore deposit is about N. 55° W. The formation outcrops for over a mile and cinnabar has been exposed at intervals over this entire distance by cuts and shallow holes. In the mine proper ore has been exposed along the strike for over 1000 ft. The ground being worked up to the date of this writing is intensely shattered.

There are two lines of croppings, one on each slope of the mine ridge. On the northeast slope the so-called No. 3 tunnel fissure outcrops and on the southwest slope the contact ore crops out. The mud fissure is a fault characterized by a heavy gouge and considerable vertical displacement. The mud fissure and No. 3 tunnel fissure cross each other on the strike as well as dip and at this juncture the ground is greatly contorted and broken. Altered igneous rocks are present in the mine in the form of dikes having a strike parallel to the main fissure system. Both the mud fissure and the No. 3 tunnel fissure are mineralized. Southwest of these is another fissure approaching the No. 3 fissure in depth, which seems to be the feeder for the contact ore. On the southwest face of the hill is another mineralized fault fissure. Many cross faults and fissures are found both underground and on the surface. Three of these faults plainly visible on the surface are denoted on the plan by a double line.

Orebodies have found under the shale capping the lime similar to those at the Nevada mine. The sand-limestone contact has ore in places. The No. 3 tunnel fissure and the mud fissure contain orebodies and there is an irregular fracture through the limestone outcropping near the top of the southwest slope of the ridge, which has been stoped extensively.

Apparently the source of the ore in this mine was a magma from which a dike came up into the shattered rock mass. The ore-bearing solutions followed the dike fracture up into the shattered zone and were confined to it by the impervious shales above. Viewed broadly the potential ore-bearing ground near this mine has hardly been touched and the combination of a greatly shattered zone, cut by intrusives and capped by shale, augurs well for future development.

Ruby Claims

The Ruby group of claims lies about 55 miles south of Battle Mountain, Nevada, and perhaps 18 miles west of the Nevada Central R. R. in what was locally called Jersey Valley. At this prospect three rock formations are of interest. The lowest formation of undetermined but great thickness is a silica cemented conglomerate coarser at the bottom than near the top. The cementation is strong, the rock breaking across the boulders and pebbles. Conformably on this conglomerate lies a fine-grained light gray to white sandstone 50 to 100 ft. thick. On this in turn lies a completely silicified fossiliferous limestone from 5 to 25 ft. thick at the outcrop.

The strike of the beds is approximately north and south and the dip is 20° W. The outcrop pitches some 8° to the south. A great fault cuts across the beds in a roughly north and south direction, as indicated in Fig. 8c.

The discovery was made by prospectors who built a fire on the outcrop and noticed globules of quicksilver in the ashes. They promptly located the ground and sank two shafts of 75 and 20 ft. respectively but struck nothing more than colors. Then they sank a shaft on the sand-limestone contact and exposed enough ore to sell the prospect. The buyer erected a retort but the ore soon ran out.

A geological study was then made of the prospect, which disclosed that the sandstone outcrop showed mineralized vertical fissures varying in thickness from knife edge to 1 in. The silicified limestone seemed very dense compared to the sandstone and so constituted an impervious capping. The fault and cross fractures from it seemed the most logical assumption for a source of mineralizing solutions, so tunnels were run from the sandstone face toward the sand-limestone contact. At the contact the limestone showed evidence of leaching and redeposition of silica as druses of quartz and chalcedony. The leached parts contained cinnabar in large massive form. Fractures in the limestone as well as the sandstone near the contact contained ore. Various runs of the retort returned yields that varied from 1 to 8 per cent. quicksilver on the weight of ore charged. This ore was found by exploring a favorable stratigraphy and not by following exposed ore, as detrital material effectively screened the vertical fissures in the sandstone.

This prospect has not been operated recently. It is isolated and water must be hauled about 8 miles. The district is interesting, however, because cinnabar is found in leached pockets in the conglomerate over a wide area. These occurrences are apparently near cross fissures from the large fault. Following the fault north some 1½ miles from the point where the sandstone and limestone are cut off, these two formations appear again on the east of the fault line.

B. and B. Quicksilver Mine

This mine lies in Esmeralda County and is reached by way of Basalt in Mineral County, Nevada. The distance from Basalt is 13½ miles, 6 miles along the Fish Lake Valley road and 7½ miles over a road built by the company. On the White Mountain, California-Nevada topographic sheet the mine would be on the Mt. Diablo base line at the letter *a* of Diablo. This location is obtained from bearings on Montgomery Peak and White Mountain Peak.

The rock in which the ore occurs is apparently a volcanic breccia or tuff, which has been intruded by igneous flows. Hydrothermal solutions from these intrusives have silicified the breccia and deposited

cinnabar in the interstices. Apparently the ore deposition was fairly recent; that is, the topography was probably not greatly different from its present appearance. Instead of concentrating under a cap rock the mineralizing solutions here seem to have spread out along the surface. The rapid cooling and large evaporation in this wind-swept location no doubt produced a concentration of the solutions akin to but milder than that caused by a confining cap rock. This supposition is borne out by the fact that near the surface the strata are of distinctly higher grade than lower down. Some fissures occur in the breccia and on some of these local concentrations of ore are found. The mass is being mined by glory-hole methods and will average 6 lb. per ton. At present prices for quicksilver and because it can be mined by cheap methods this deposit is ore. It illustrates the importance of a confining cap rock by showing the type of deposit formed where no such cap rock existed.

The top of the hill is mineralized over an area of about 400 by 600 ft. The general strike of the ore zone is northeast and southwest and there seem to be two systems of fractures through the mass. The cementation of the breccia is so strong that the rock breaks across the pebbles. Aside from trenches and short tunnels near the present glory hole, little exploratory work has been carried on, though cinnabar outcrops down the hill slope under the furnace and camp sites.

Castle Peak

The Castle Peak deposit is still in the development stage but since it represents a distinct type of deposit it merits a brief description. It is reached by following the Virginia City-Steamboat Springs road to the Five Mile House and following the road which branches off to the north at this point. On the Carson-Nevada topographic sheet it can be located by drawing a line ½ in. east of the 119° 40′ line of longitude and another line 2½ in. north of the 39° 20′ line of latitude. The intersection of the lines so drawn locates the deposit.

An altered andesite outcrops here on a north to northeast strike. This outcrop varies from 20 to 80 ft. in width and has been trenched for a length of over 1000 ft. The dip and the wall rocks have not so far been determined. Limestone is found on the west but the relation between the two rocks is not yet apparent. A dike of an altered intrusive cuts the andesite and a number of fractures and fissures are exposed in the workings. The entire mass of the andesite seems to be more or less mineralized. At the southern end, where the dike cuts across, a fair concentration of mineral has occurred in fractures and brecciated areas. In a general way the deposit resembles the Black Butte, Non Pareil and Bonanza mines of Oregon, which will be referred to later on.

A 3 by 40-ft. rotary furnace plant is being erected (July, 1929) and is expected to be in operation in August, 1929.

Steamboat Springs

Steamboat Springs has been described and mapped by Becker[57] Here, as at the B. and B. mine, a cap rock is absent. The ore-bearing solutions came up through granite. Of the granite only the silica remains, in clear grains resembling glass sand under the microscope, with here and there a grain of cinnabar. Here also there is a richer surface layer. A large area of the granite is covered by basalt. At the northern edge of this basalt sheet the granite has been decomposed by solfataric action and cinnabar has been deposited in it. It would seem that concentrations of cinnabar might be found in decomposed granite under the basalt cap similar to the occurrence of ore at Sulphur Bank, Knoxville and Oat Hill in California.

Other Occurrences in Nevada

The cinnabar deposits near Mina have been described by Knopf[58] and Foshag.[59] From their descriptions it is evident that production so far has come from small local concentrations of cinnabar, and that thus far no area favoring a concentration of the primary mineralization at some definite horizon or plane has been found.

Many quicksilver properties are being developed in Nevada at the time of this writing. At present the Antelope Springs district seems the most promising. Here there are fractured and intruded limestones and sandstones capped by impervious shales that confine the ore-bearing solutions to definite horizons and induce a concentration of the primary mineralization in minable orebodies. Note that none of the Nevada deposits described exhibit such favorable conditions for primary concentration as do many of the California and Texas deposits. None of the Nevada mines have approached these in production and the relation between production and stratigraphy favoring primary concentration is more than a mere coincidence.

Quicksilver in Oregon

Two deposits furnished the major production of Oregon for 1928. Both are remarkable. Prospecting and development are active in the state and an increased production for 1929 is probable.

Opalite

The Opalite mine is 20 miles from McDermitt, in the southeastern part of the state. The geology of the region has not been studied or

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[57] G. F. Becker: *Op. cit.*, text and atlas sheet XIV.

[58] A. Knopf: Some Cinnabar Deposits in Western Nevada. U. S. Geol. Survey *Bull.* 620 (1915) 59.

[59] W. F. Foshag: Quicksilver Deposits of the Pilot Mountains, Mineral County, Nevada. U. S. Geol. Survey *Bull.* 795 (1927) 113.

described, except in a general way by Russell,[60] who mapped the many major fractures of the region. The mine croppings are a prominent

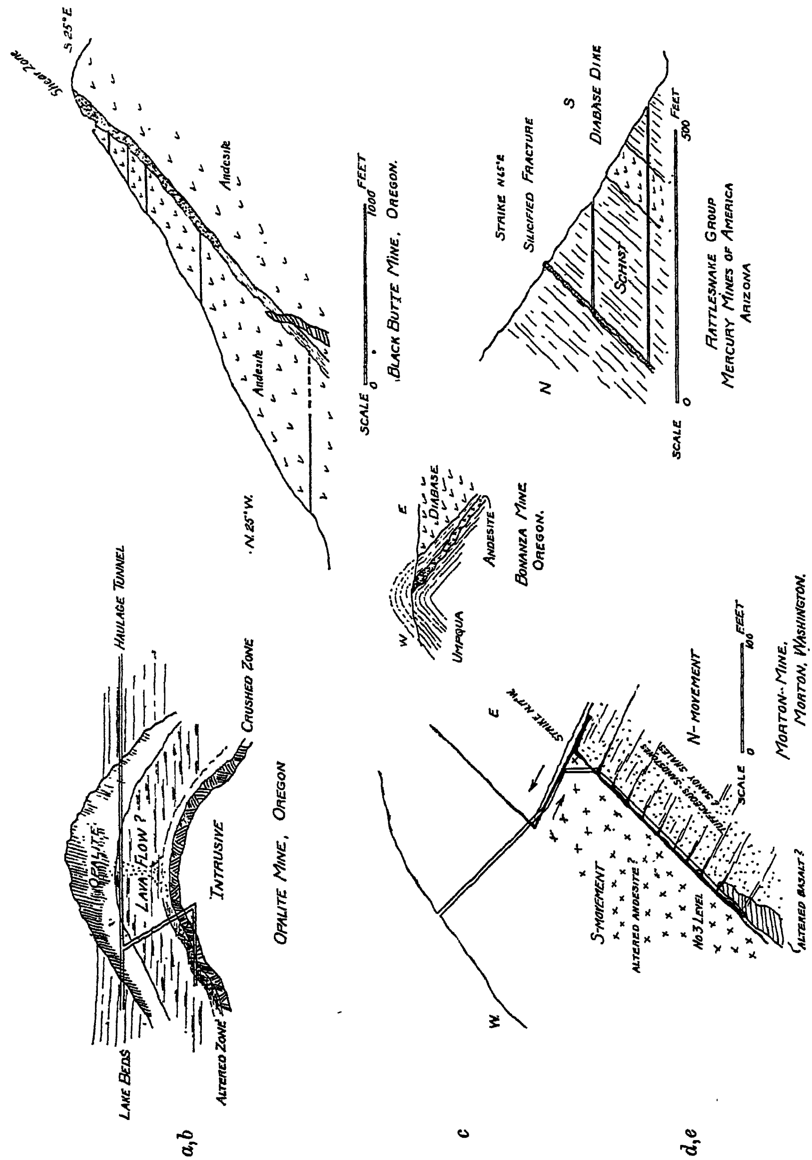

FIG. 9.—QUICKSILVER DEPOSITS IN OREGON, WASHINGTON AND ARIZONA.

feature of the landscape. The rock is discolored on exposure to the light, and cinnabar is visible in the outcrop only when fresh surfaces are broken

[60] I. C. Russell: A Geological Reconnaissance in Southern Oregon. U. S. Geol. Survey 4th Ann. Rept. (1882–83) 431–464.

off. This is probably the reason why it was not discovered until 1924, though quicksilver was known to occur in this region long before that time.

A section through the deposit is given in Fig. 9*a*. The lowest rock is a coarse-textured intrusive with a crushed zone some 8 to 10 ft. thick along its upper surface. Above this intrusive is a dense fine-grained rock, which shows what may be either stratification or flow structure and is sprinkled with minute pyrite crystals. Along the contact with the crushed zone of the underlying intrusive it has undergone alteration, and some cinnabar is found in this altered zone. Above this rock is a deposit of opalite or siliceous sinter, evidently deposited from siliceous springs. The upper portion of this siliceous opalized sinter contains cinnabar and constitutes the orebody. After the orebody was formed, lake beds and tuffs were deposited, in which many specimens of petrified wood have been found. The deposit is similar in many ways to the B. and B. mine, except that here the alkaline silica-bearing solution formed a sinter mound while at the B. and B. the deposition took place in a tuff or breccia lying on the surface at the point of emission. As the B. and B. also, the deposit as a whole is low in grade, as no concentration of the deposition took place other than that probably provided by evaporation when the mineral-bearing solutions reached the surface. There is geological evidence that little or no erosion has occurred at deposits of this type.

The mineralized area exposed on the surface measured some 400 by 250 ft. The lower surface of the ore is very irregular, and the average thickness of ore varies between 30 and 50 ft. Thus far no definite stock or channel has been found to show where the solutions came up, and it is not clear whether the mineralizing solutions came directly from the lower intrusive or up through the altered zone lying along its upper surface.

Other quicksilver prospects are known in the region. Disaster Peak, an old volcano, lying only 9 miles distant and hot springs, about 3 miles from the mine, give evidence of comparatively recent volcanic activity.

Black Butte Mine

This quicksilver mine is 17½ miles south from Cottage Grove, Oregon, in T. 23 S., R. 3 W. Other quicksilver deposits in this general region are at Elkhead, 7 miles southwest from Black Butte, the Non Pareil and Bonanza, about 9 miles directly south of Elkhead, and a prospect at Glide, 20 miles south of Black Butte. The mine is in an east-west spur of a ridge on the north slope of the Calapooia Mountains, a range connecting the Cascades and Coast ranges. All exposed rocks in the vicinity are volcanic,[61] mainly ash rocks, and a few massive lavas. The volcanics have undergone alteration. Thermal waters have silicified what appears

[61] Data from an unpublished report by H. W. Turner, 1902.

to have been andesites, and on account of this silicification they have withstood erosion and so formed the Butte.

The ore deposit lies in a tremendous fracture zone between two clean-cut fractures that form the walls. The footwall particularly is remarkably smooth, hard and persistent. The strike of the fracture zone, which was formed after silicification, is S. 67° E. and the dip varies from 50° to 60° NW. Basalt intruded the andesite and is in part mineralized and silicified. The mineralization then probably followed close upon the basalt intrusive along the same planes of weakness followed by the basalt. The mineralizing solutions rose through the shattered mass of altered silicified andesite between the two wall fractures. The walls are from 20 to 40 ft. apart and the fracture zone is 3500 ft. long. The main mine workings to date extend some 1500 ft. along the strike. The fractured rock in this tremendous space presented no great pore space but only thin narrow seamlets for ore deposition, so that no concentration into high-grade orebodies could take place. The ore zone dips with the hill slope, so that comparatively short tunnels serve to reach it at depth. This fact and the strength of the walls, which permits large open stopes in the mining operations, have much to do with the classification of this mineral occurrence as ore. A section of the deposit is given in Fig. 9*b*. The ore in this mine, as in others previously described, carries gold. Assays up to $2.17 per ton have been obtained. The ore is distributed along the strike and dip with fair regularity and at present prices of quicksilver ($120 per flask) practically the entire content of the fracture zone can be stoped. The low-grade content of the large ore deposit illustrates the importance of voids under the confining rocks, if the primary mineralization is to be concentrated. Here the space for deposition is so meager that, despite the large amount brought up, the cinnabar is so thinly deposited that it constitutes an orebody only because of the rise in the price of quicksilver and because low-cost methods of mining can be used.

Bonanza and Non Pareil

These two mines, discovered in 1865, are about 8 miles east of Sutherlin, Oregon. They are shown on the Economic Geology Sheet of *Folio* 49 of the U. S. Geological Survey, 6 miles slightly south of the town of Oakland. The northern mine is the Non Pareil and the southern one, a little west of it, the Bonanza. Apparently both lie on the same dike of a greatly altered andesite, which parallels the diabase intrusion east of it. The dike of altered ore-bearing rock strikes about N. 20° E. at the Bonanza and N. 35° E. at the Non Pareil. The dip varied between 45° at the Bonanza to 65° E. at the other. The intrusion apparently came up in the stratification of the eastern leg of an anticline in the Umpqua formation. This suggests that the later andesite came up along the plane of weakness along the diabase-Umpqua contact, and then

pushed out along the stratification as indicated in the sketch, Fig. 9*c*. The hanging wall is a shale, while the footwall is a hard dense sandstone. The dike itself varies in thickness from 30 to 90 ft. and is mineralized to some extent throughout. The cinnabar is not easily visible to the eye but panning the decomposed material gives surprising results. This dike is not silicified, but shows evidence of dolomitization. Here, as at Black Butte and at Castle Peak, Nevada, there was little pore space in the rock, and except locally under fault gouges and in cross fractures there was no concentration of the mineralization and only a low-grade orebody resulted. The ore-bearing dike seems to parallel the diabase intrusion some 7 miles, and other prospects are known south of the Bonanza. It may be that the part of the dike now exposed merely acted as a feeder to an orebody which formerly lay higher up under the crest of the anticline in the sands and shales of the Umpqua formation. It is further possible that uneroded remnants of this anticline remain along the strike of the andesite dike and that here concentrations of the primary mineralization await exploitation by the miner. It is from such prospecting that future ore supplies must be sought.

Quicksilver in Washington

There is a quicksilver-mining district around Morton, some 67 miles south of Tacoma, Washington. The two mines active in 1928 were the Morton Cinnabar Co. and the Barnum McDonnell mines. The properties adjoin and are on the same ore occurrence. The mineralized area, of which these two mines form the center, extends about 3 miles NW.–SE. and perhaps 2 miles NE.–SW. The rocks in the district are feldspathic tuffaceous sandstones and sandy shales containing organic matter. Some of the rocks are derived from pyroclastics, such as volcanic ash and breccias. These country rocks were intruded by andesite and basalt now greatly altered.

The strike of the ore occurrence is N. 17° to 18° W. and the dip is from 26° to 45° W. A section through the Morton company's ground is given in Fig. 9*d*. No. 3 level is the main haulage level and is run on the contact of an altered andesite (?) and a tuffaceous sandstone-shale series. There has been sufficient movement along this contact to make a gouge, and apparently the movement was north in the east and south in the west sides of the contact. In the upper workings a thrust fault has displaced the contact some 60 or 70 ft. toward the west. A gouge of black sandy material 4 in. thick was formed along this fault plane. The ore has formed under these gouges, which constitute the hanging wall. Toward the footwall the ore grades off. Counting both mines, ore has been developed on the strike for a distance of over 800 ft. Locally, where favorable conditions for concentration are found under shale strata, high-grade pockets are formed. One mine stopes a furnace ore, the other

selects retort ore. In depth, ore has been found down to 400 ft. thus far. The McDonnell mine on its lower level has a small seam of coal interstratified with the sedimentaries. At this point, cinnabar, pyrite and coal occur together. Here is an opportunity for geologists who advocate a magmatic origin of bituminous matter in quicksilver deposits to enlarge upon their theory and include coal deposits.

On the Morton company's ground a NE.-SW. fracture dipping south has been worked. Toward the east on the strike of these workings a dike shows on the surface. The intersection of this fracture with the fault on which the main workings are located is looked upon as promising ground.

Quicksilver Deposits in Arizona

The geology of the quicksilver deposits of Arizona has been described by Gardner and Lausen,[62] by Ransome[63] and by Schrader.[64]

Only the mines in the Mazatzal Range were productive in 1928. Two properties in this district are equipped with furnaces. The first property visited was that of the Mercury Mines of America, Inc., 73 miles north from Globe, past the Roosevelt Dam. On the Roosevelt quadrangle topographic sheet the property can be located in the northwest corner of the sheet, at the letter "l" of the words "Slate Creek." The other working property is that of the Arizona Quicksilver Consolidated, about 5 miles west on Alder Creek.

Mercury Mines of America has the Ord group of claims, and through a subsidary known as the Tonto Mining Co. also controls the Red Bird or Arizona Cinnabar, Rattlesnake, Pine Mountain, Mercuria and Boardman group of claims.

Arizona Quicksilver controls the Sunflower, Robbins, Big Four Consolidated and L and N groups, together with 42 mill sites.

The geology of these deposits has been adequately discussed by competent geologists and it remains to speak only of the ore occurrence. This is similar in all the deposits thus far found, both in the Mazatzal Range and in the Phoenix Mountains. The rocks in which the ore is found are schists interstratified with intrusives. Schists, even when as well fractured as these, are not conducive to the formation of brecciated areas giving sufficient interstitial space for the concentration of large orebodies. There are rock strata in the district that would make splendid receptacle rocks for ore, but unfortunately the schists are the

[62] E. D. Gardner and C. Lausen: Quicksilver Resources of Arizona. Univ. Ariz. *Bull.* 122 (1927).

[63] F. L. Ransome: Quicksilver Deposits of the Mazatzal Range, Arizona. U. S. Geol. Survey *Bull.* 620 (1915).

[64] F. C. Schrader: Quicksilver Deposits of the Phoenix Mountains, Arizona. U. S. Geol. Survey *Bull.* 690 (1918).

weakest rocks and most of the fracturing took place in them. All the ore-bearing fractures in the schist generally run with the strike of the beds and reach the surface, and nowhere on the deposits thus far found is there a flat-lying cap rock under which orebodies could have formed by concentration of the primary mineralization. The mining then is confined to these fractures in the schist, which are filled mainly with quartz and a little cinnabar. Most of these fractures are narrow, so that a good deal of waste must be broken with the ore. In some of the fractures the schists are contorted and broken, and at such places fair orebodies up to 4 ft. wide are found. A section through one of the fractures is shown in Fig. 9*e*. At the Arizona Quicksilver Consolidated mine the main fracture is on the Packover claim. This strikes east of north and is offset by a cross fault for 150 ft. The ore is found on both sides of the fault in a fractured zone in the schist, some 50 ft. wide. The best ore is found near the fault, where fracturing was greatest. The fault is also mineralized, showing that it is premineral. The ore has been followed down to a depth of 240 ft. with the Packover shaft. Little flat streaks of gouge through the shattered schist caused local concentrations of cinnabar under them, which helped greatly in making an orebody at this point.

As thus far developed the quicksilver deposits found in the schists have not a structure favorable to the formation of large high-grade orebodies. Favorable conditions for ore deposition exist locally in the fractures and fracture zones. These scattered in-and-out concentrations of cinnabar yield the ore being mined. Generally the richer they are the less there is of them.

Mineralization was evidently extensive over a large area. While most of the fracturing took place in the weaker schists, there are probably also fractured limestones and sandstones in this district. These should be carefully prospected for favorable structures, as these rocks if mineralized offer better chances of finding cinnabar orebodies than do the schists.

Quicksilver Deposits in Mexico

Mexico, though a great consumer of quicksilver, particularly in the heyday of the patio process of silver amalgamation, has been a comparatively small producer of the metal. This is somewhat surprising when it is considered that a great demand existed, that tax exemptions to quicksilver mines and bonuses were offered as inducements to home production, and that cinnabar has been found in Mexico in localities too numerous to mention. Garcia[65] lists some 50 localities in 13 of the Mexican states as cinnabar bearing.

[65] J. A. Garcia: El Renacimiento de la Industria Minera del Mercurio. Mexico, Dept. de Minas, *Bol. Min.* (Jan., 1927) **23**, No. 1, 34.

Humboldt[66] visited and described many quicksilver deposits in what is now Guanajuato, Zacatecas, San Luis Potosi and Guerrero. His visit and the account thereof date in the early years of the nineteenth century. He states that while few countries have so many indications of cinnabar as the tableland of the Cordilleras from the 19th to the 22d degree of north latitude, the work on these deposits has been frequently interrupted, conducted with little zeal and generally with little intelligence. He cites these facts as being responsible for the undeveloped state of quicksilver mining and shows the importance of the quicksilver production by stating that the annual production of silver depended at that time mainly on the facility with which the mines procured the quicksilver necessary for amalgamation.

The literature relating to the quicksilver mines of Mexico is not extensive and the description of most of the orebodies is meager. Detailed maps and geological sections apparently have not been published anywhere. Such descriptions as are available indicate that the Mexican orebodies were deposited in conformity with the theory of ore deposits formed by primary concentration.

Humboldt described a quicksilver deposit at Durazno between Tierra Nueva and San Luis de la Paz, in what is now the State of San Luis Potosi. This deposit was worked through shafts 15 to 20 ft. deep. It was a flat low-grade deposit of great horizontal extent on the contact of porphyry and clay slates. The mine was lost by caving at a later date, due to insufficient support of the flat clay-slate roof. Meager as this description is, it does indicate that a concentration of the primary mineralization under the impervious clay slates was responsible for the formation of the orebody.

Dulces Nombres Mine

Babb[67] describes the Dulces Nombres quicksilver deposit in the State of San Luis Potosi. Dulces Nombres Mountain, in which the mine lies, is approximately 15 km. west of the railroad at Enramada, a station on the Mexico-Laredo line of the National Railroad.

The formations in this area are Cretaceous sedimentaries, which have been intruded by igneous rocks. The mine is in an uplifted area 4 km. long in a north and south direction and about 3 km. wide. Two large arroyos dissect this mountain mass. The main body of the mountain is composed of stratified limestones of the Lower Cretaceous, which are distinguishable by their black to gray color, from contained bituminous matter. Middle Cretaceous limestones, lighter in color and interbanded with chert, rest on the Lower Cretaceous strata. Several hundred meters

[66] A. Humboldt: Essai Politique sur le Royaume de la Nouvelle Espagne. Paris, 1811. Schoell.

[67] P. A. Babb: Dulces Nombres Quicksilver Deposit, Mexico. *Eng. & Min. Jnl.* (Oct. 2, 1909) 684.

to the southeast of Dulces Nombres, upper Jurassic calcareous sandstones apparently are faulted against the Cretaceous formations.

The dip of the limestones is some 50° W. While no igneous rocks outcrop on Dulces Nombres Mountain itself, the neighboring mountains

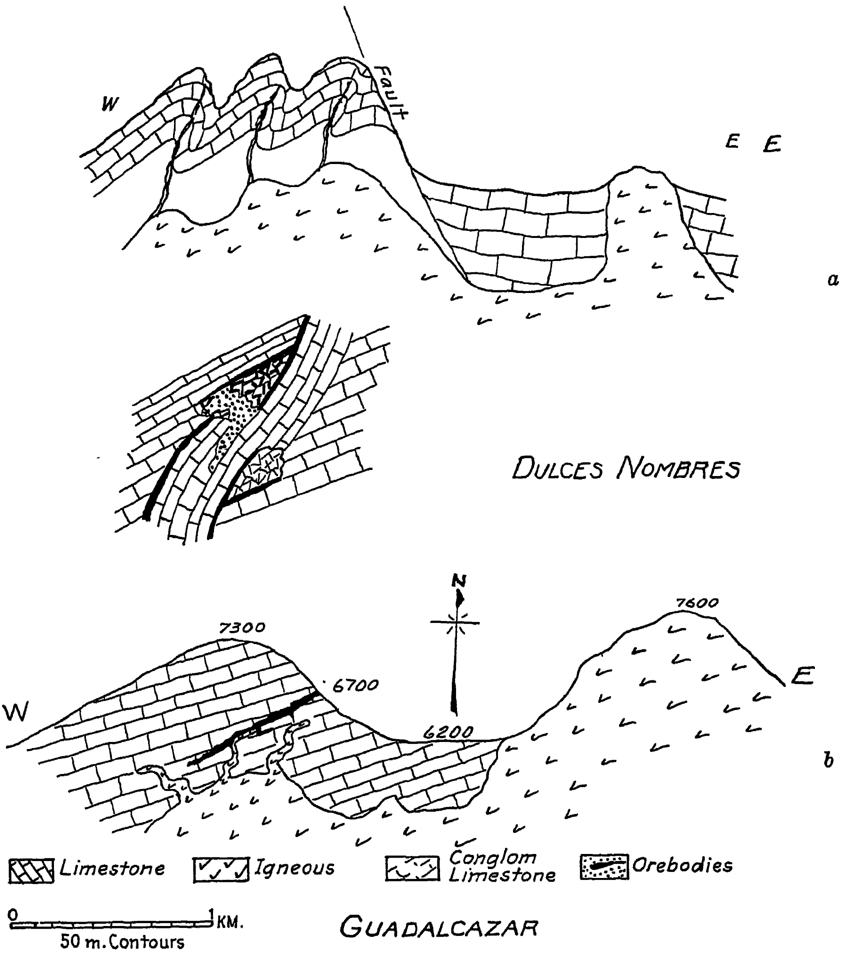

FIG. 10.—QUICKSILVER DEPOSITS IN MEXICO.

to the northeast and south show igneous rock and lava flows. A few kilometers to the east are several eruptive cones, and it appears probable that Dulces Nombres Mountain was uplifted by a laccolite below it.

A formerly active magma as a probable source of quicksilver is therefore indicated. At the east side of the mountain a great fault marks the shearing of the uplifted mass from flat-lying beds to the east. Major

fractures through which magmatic solutions could rise then are also present. Parallel planes of weakness developed in the uplifted rock mass and Z-shaped folds are formed in the strata. Many small irregular and distorted areas of Z-shaped section were formed. Many of these were ruptured, gouges and slickensides indicating that considerable movement took place. The rock in the acute angles of the Z folds was brecciated. This condition is illustrated in Fig. 10*a*, after Babb.

The ore is found under the gouges within the *downward opening* acute angles of the folds. This is an excellent sample of a trap in which the primary mineralization was concentrated, the more so as the upward-opening angles of the two folds *are never mineralized.* The deposit in a general way apparently is very similar to the deposits of the Big Bend region in Texas, particularly the Mariscal mine, where cinnabar is also in part found concentrated in local folds in the limestone. There, as here, cinnabar, calcite, a little pyrite and gypsum are the main minerals associated with the deposit. A little native quicksilver is found in the Dulces Nombres deposit, which is not strange in view of the abundant organic matter reported in the limestone. Several rare quicksilver minerals are reported in one part of the mine and here again the similarity to the Texas deposits is apparent.

This ore occurrence illustrates the necessity of trapping the mineral-bearing solutions for the formation of an orebody by concentration of the primary deposition. In this mine the ore-bearing solutions rose through brecciated rock along the planes of weakness. Part of this breccia is situated in capped folds and part in folds opening upward. *Orebodies formed only in the capped folds.*

Guadalcazar

The quicksilver mines at Guadalcazar, 70 miles northeast of the city of San Luis Potosi in the state of the same name, are described by Mactear[68] and Collins.[69] It is estimated that the mines of this district have produced between 50,000 and 100,000 flasks of quicksilver. These mines also, as far as known, are the only ones to earn the premium offered by the government for a yearly production of 2000 or more quintales (2700 flasks) of quicksilver.

The quicksilver deposits occur along a limestone ridge for a distance of about 2 miles. This ridge trends northwest and southeast, the strata dipping gently southwest. The ridge takes the form of a series of rounded hills, many of which are capped with gypsum deposited from mineral springs. These limestones have been intruded by flows of

[68] J. Mactear: Mining and Metallurgy of Quicksilver in Mexico. *Trans.* Inst. Min. and Met. (1895–96) 4, 69.

[69] H. F. Collins: Quicksilver Mining in the District of Guadalcazar, State of San Luis Potosi, Mexico. *Idem,* 121.

andesite and a parallel ridge of granite outcrops to the east. Dikes of altered granite are found in the mines. Caves are common in the limestone and some of the limestone beds have been altered to gypsum. Some of the caves have broken through to the surface. The rocks encountered in the exploration and exploitation of these mines are the following:

1. *Unaltered Limestone.*—This is hard and bluish in color. It contains 8 to 12 per cent. insoluble matter, mostly clay and some free silica. Near the quicksilver orebodies the organic matter of the limestone is discolored.

2. *Breccia.*—Small fragments of generally siliceous limestone, sometimes loosely cemented by calcite and gypsum but oftener by clay. Two varieties of this breccia are distinguished, one of them containing so much clay as to be practically impervious to the passage of solutions. One mass of such rock in the Trinidad mine is over 200 meters long, 50 meters wide and of unknown depth.

3. *Altered Granite Dikes.*—These are thought to furnish part of the clay cementing the breccia, the rest of the clay being the insoluble part of the limestone.

4. *Gypsum.*—This occurs in large distinctly bedded rock masses of the same dip and strike at the limestone beds with which it is interstratified. The gypsum has been formed by alteration from limestone. These gypsum beds were probably formed from the limestones by acid waters of volcanic origin at a time preceding the ore-bearing solutions. In the change from limestone to gypsum there is an increase in volume of more than one-half. This excess or part of it seems to have been carried off by the solutions effecting the change and to have been deposited at the surface.

5. *Rock Resembling Bone Ash,* which fills the small fissures connected with the orebodies.

The minerals found in the orebody are cinnabar, metacinnabar and two sulfoselenides, probably guadalcazarite and onofrite.

The cinnabar occurs in two different manners, as crystalline cinnabar in fissures of the wall rock near the orebodies and as dull red powdery ore scattered through the larger orebodies and in stringers in the gypsum wall rock. The ore minerals are scattered through the breccias that constitute the orebodies.

There are several ore occurrences along the ridge, which are worked as separate mines.

San Antonio Mine

This mine is entirely in limestone. The ore is found in a breccia of limestone fragments and clay. The strike of the orebody is north and south and the dip in general is west, varying from flat to vertical.

The ore minerals are cinnabar and metacinnabar. The ore-bearing breccia extends 15 to 25 meters along the strike and varies from 2 to 6, sometimes 9 meters, across the strike. The horizontal section of the orebody is irregular.

Apparently there was here a steplike fissure through bedded limestone, sometimes breaking across the beds and sometimes along the bedding plane. Magmatic waters probably following the granite or andesite intrusion rising in this irregular fissure enlarged it by dissolution of the walls and deposited the dissolved matter at the surface. The enlarged fissures became loosely filled with caved portions off the walls and the insoluble clay of the limestone. This porous or spongy mass served to trap or arrest the probably alkaline solutions that came up later and deposited their load of mineral matter, *i. e.*, cinnabar. A recurrent flow of acid magmatic waters at a later time, which seems probable in view of the gypsum mounds at the surface of some of the fissures, would account for the change of a part of the cinnabar to the black sulfide. Additional evidence on this point will be brought out in discussing the San Antonio de Padua mine.

The ore occurrence at the San José mine is similar though on a smaller scale, the orebody being more nearly described as a thin sheet.

El Quiote and Santa Maria

Here there are three mines, the Guadalupe, Angustias and Las Animas on the same fissure. This fissure, dipping steeply to the south, runs east and west at right angles to the strike of the other deposits. The fracturing is entirely in limestone though a decomposed andesite outcrops near the fissure. The hanging wall is smooth and distinct but there may be several false footwalls, especially where the fissure widens out. The fissure filling is altered limestone from the walls, clay and crystallized calcite. There is a layer of clay from 1 in. to 2 ft. thick on the hanging wall in bands of varying color. The banding follows the configuration of the wall. The ore is cinnabar. This mineral is scattered through the breccia filling the fissure and also forms a thin but constant stringer just under the clay on the hanging wall. The fissure is 3 to 4 ft. wide. Las Animas pinched out and became barren in depth and the fissure was lost in a cavern under it.

These mines lie on what was probably originally a fault fissure with a thin attrition gouge. Magmatic waters rising through the fissure enlarged it by solution. Part of the residual clay was deposited along the hanging wall by the rising waters, differences in the solution or recurrent solutions accounting for the color bands. Sloughing of parts of the steep footwall as it was undercut by solutions penetrating cross fissures or bedding planes would explain the false footwalls in wider parts of the fissure which became filled with clay and wall fragments.

After this porous rock mass, containing many small individual pockets or traps, had been formed, the mineralizing solutions rose and deposited their load in these traps and along the hanging wall under the impervious clay. The absence of metacinnabar in these mines conforms to the absence of gypsum beds. Evidently acid solutions did not traverse these rocks to any extent before or after the ore deposition.

San Antonio de Padua

Collins says that this mine, although the newest, was the most productive mine to that date (1895) and by far the most extensively worked quicksilver mine in the state. The outcrop was discovered on the side of Ardillas Hill, the ore dipping into the hill 15° to 20°. This hill, unlike the others in the vicinity, is made up of interstratified beds of limestone and gypsum and irregular masses of the latter (Fig. 10*b*).

The orebodies, containing a good deal of metacinnabar, are irregular in form and are arranged along a series of fault fissures nearly parallel to the stratification. Breccias are formed along the entire fault zone. The largest orebodies are formed under these breccias and where these are of the very clayey type they rarely, if ever, contain ore.

Here apparently the limestone beds were faulted along the bedding plane with enough movement to form a fault gouge. Acid waters of volcanic origin rose through this fissure and enlarged it by solution. The footwall under the fault gouge is generally gypsum and this bed may have been formed at this time also. The breccia (No. 2 of above description) typical of this district now formed under the gouge created by movement along the fault fissure. Part of the gypsum formed was removed in solution to the surface and the solution cavity was filled by a mixture of residual clay and wall fragments.

At a later date, perhaps connected with the andesite intrusion, alkaline magmatic waters carrying quicksilver in solution rose through this brecciated zone and deposited cinnabar in the traps formed by the voids in the breccia.

In the footwall under the ore a feeder fissure filled with the rock resembling bone ash (No. 5 above) was discovered. The description of this fissure, which is never quite barren of cinnabar, reminds one strongly of the Elvan streak as disclosed in the lower workings of the New Idria mine and it is suggested that it represents a strongly altered intrusive dike of the ore-source magma. It may have intruded after a renewed movement along the plane of weakness followed by the original fault.

Where this fissure cuts steeply through the gypsum it is not wide, nor does it carry much quicksilver, but where it turns from the vertical and follows the beds—where rising solutions following its course were checked —orebodies conformable to the stratification were often developed.

At some time after the cinnabar had been deposited by solutions intimately connected with this intrusive, renewed volcanic activity resulted in a recurrence of acid waters which changed part of the ore to metacinnabar as it is now found.

Lest it seem that the facts are being strained to fit a theory, a review of the evidence is in order.

First we have interstratified series of slightly dolomitic limestones and gypsum beds. It seems reasonable to conclude that these gypsum beds were formed in place by alteration from the original limestone beds. The most probable agent to accomplish this result would be an

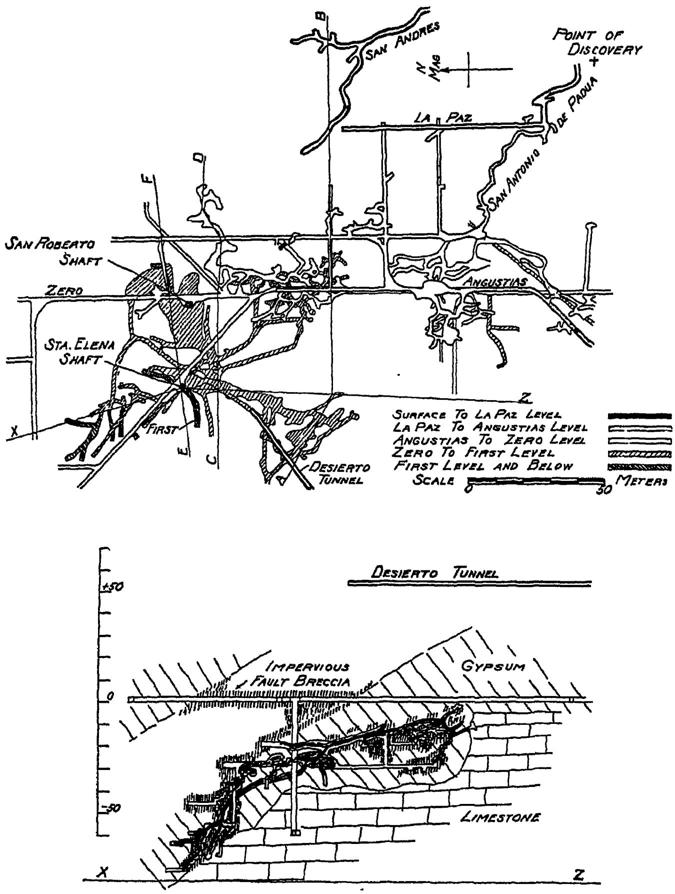

FIG. 11*a*.—SAN ANTONIO DE PADUA MINE.

acid water of volcanic origin. The large solution caverns filled in part with the insoluble clay of the limestone and the clearly shown interstratification all speak for this conclusion.

The altered intrusive in intimate association with the ore deposits and the close parallel it affords with the Elvan streak of New Idria speaks for the usual agencies, *i. e.*, alkaline solutions having brought

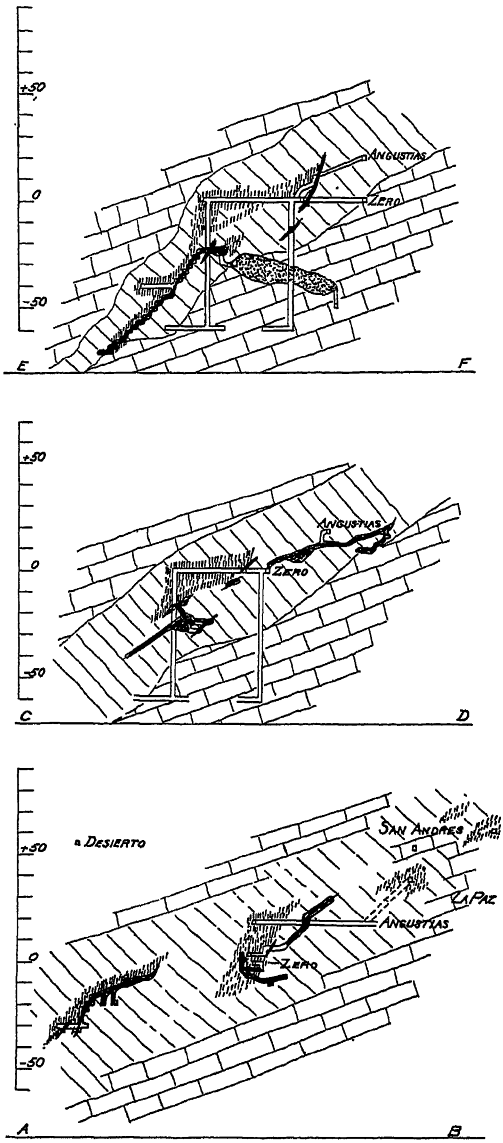

FIG. 11*b*.—SAN ANTONIO DE PADUA MINE.

in primary cinnabar in the usual way. The fracture through which this intrusive and the alkaline solutions rose cuts the gypsum and therefore followed the first acid phase.

Evidence of a recurrent phase of acid solution is given by the gypsum mounds on the present surface though if the ore deposition was relatively recent these may have been partly formed during the original change of the limestone to gypsum.

The occurrence of the cinnabar in two forms is an argument for a recurrent acid phase. Crystalline cinnabar is found in the joint planes of the limestone at considerable distances from the main mass of the orebody. Such crystalline cinnabar would be primary and deposited from the alkaline solutions. During the recurrent acid phase these crystals were not reached by the acid waters, on account of their reaction with the lime before they penetrated so far along the fissure. The recurrent acid phase apparently was not as prolonged as the first one. The dull red powdery cinnabar on the main orebodies and in fissures in gypsum (contrast with crystalline cinnabar in fissures in limestone) may be secondary, as the metacinnabar. Practically it makes no difference, as explained in the presentation of the theory, as it has not migrated to any appreciable extent and is present at the same place and in the same quantity as when originally deposited as a concentration of primary cinnabar.

If one would argue that the secondary minerals were formed by acid surface waters subsequent to the deposition of cinnabar, the absence of metacinnabar in the Guadalupe, Angustias and Las Animas mines described above must be accounted for by supposing that surface waters did not enter these orebodies. The country is dry, there is no water in the mines, and the gypsum mounds on the surface argue for a recurrence of acid volcanic waters to account for the secondary minerals in the mines.

The mine is illustrated on Fig. 10*b* and Fig. 11*a* and *b*.

Quicksilver in Durango and Guerrero

Villarello[70] describes a deposit at Palomas, 80 km. west of Durango in the State of Durango. This deposit lies in a fissure in rhyolite close to a contact between the rhyolite and a later basalt. The rhyolite is bleached and altered on the contact, probably by magmatic solutions. This alteration extends from the contact to the fissure. The fissure is filled with amorphous silica, sometimes colored red with cinnabar. In other places concentration of cinnabar seems to have taken place under clay gouges, at least he speaks of these ore pockets being encased by clay.

In Guerrero he describes La Cruz y Anexas mines about 1 km. south of Huitzuco. They are in Middle Cretaceous limestones. The orebody is formed by a series of great pockets aligned in depth and having a horizontal diameter of 50 to 80 meters. These pockets are filled with a ferruginous clay breccia apparently similar to those described by Collins

[70] J. D. Villarello: Génesis de los Yacimientos Mercuriales de Palomas y Huitzuco. Soc. cient. Ant. Alzate. (1903) 19, 95.

at Guadalcazar. The impregnation of the breccia with the ore minerals of cinnabar and livingstonite is irregular and rich pockets of cinnabar ore are found "wrapped up in clay which is barren."

While the description is not detailed, it seems to indicate a step fault fissure through limestone beds, enlarged by solution similar to those at Guadalcazar. Intrusive andesites are found to the northeast. The mineralizing solutions carried up antimony as well as quicksilver and pyrite. His observation that rich cinnabar is found encased in barren clay is significant evidence on the value of the trap formation in concentrating the primary mineralization.

In another paper he describes[71] the quicksilver deposits of Chiquilistlan, above 190 km. southwest of Guadalajara in the State of Jalisco. These somewhat isolated deposits are situated at the foot of a mountain range that trends southeast-northwest. Two rock formations dominate the district; namely, Middle Cretaceous sedimentaries and Tertiary eruptives. The sedimentaries are limestones covered in places by flat deposits of sandstone, clay and marl. The eruptive is a hornblende andesite. The flat-lying sandstones, clays and marls strike 45° NW. and dip 20° NE. The thickness of these cap rocks is small, as the solid limestones are found in the mines at no great depth. The limestones are fractured in various directions and while the amount of exploratory work has not been great it seems that the ore is found where two or more fracture systems cross. No great faults have been found and the structure of the limestone is monoclinal. Villarello holds these fissures to be contemporaneous with the emission of the andesite and to have served as channels for the passage of magmatic waters. He is inclined to believe that the fissures have been enlarged by solution before ore deposition. Quicksilver, copper and iron were here brought up by the magmatic solutions and were deposited and concentrated in the open fissures of the limestone under the impervious capping of the overlying clay and marl. Erosion later removed the cap rocks in whole or in part and surface water then changed the sulfides of copper and iron into the carbonates azurite, malachite and limonite as they are found in the workings. Here again in a different part of Mexico we have quicksilver orebodies formed in fractured ground, near an intrusive and under a cap rock that served to confine the mineralizing solutions.

A third paper by Villarello[72] describes the Bella Union mine 2 km. south of La Cruz y Anexas in Huitzuco, Guerrero. It is apparently very similar to the other mines, though it had not been developed to great depth.

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[71] J. D. Villarello: Descripcion de los Criaderos de Mercurio de Chiquilistlan (Jalisco). Soc. cient. Ant. Alzate. (1904) **20,** 389.

[72] J. D. Villarello: Description des mines "La Bella Union" (Etat de Guerrero). Soc. cient. Ant. Alzate. (1906) **23,** 395.

The data on quicksilver orebodies of Mexico is not nearly as extensive as one could wish but from the descriptions as reviewed above, the conclusion that they were deposited in conformity with the theory of primary concentration is permissible. Certainly the most important orebodies most nearly fulfill the conditions required, such as evidence of a formerly active magma, major fractures for the ascent of magmatic solutions and a trap formed in pervious rock under an impervious capping. The occurrence of secondary minerals in some of the deposits is explained by special conditions subsequent to the deposition of the original orebody, which was no doubt primary cinnabar.

Quicksilver Deposits in Peru

Becker[73] lists 11 localities in Peru where cinnabar is known to occur. It has been mined in several places but the most important orebodies were in the neighborhood of Huancavelica. This district is one of the notable quicksilver-producing localities of the world, with a production of over 1,500,000 flasks.

Umlauff,[74] in his description of the Huancavelica deposits, describes and has plotted on a map a series of quicksilver occurrences extending from 30 km. northeast to 30 km. southwest of the city. The country as mapped by Umlauff is made up mainly of Cretaceous limestones interbedded with shales or slates. The zonal line of the quicksilver occurrences is marked by outcrops of sandstone and shale and by small andesite intrusions. A large area of basalt outcrops southwest of Huancavelica and other large areas of porphyry outcrop northeast and southwest of the line of quicksilver mineralization. Limestone conglomerates are found along the mineralized belt and northeast and southwest of it.

Apparently the quicksilver deposits are situated along a major fracture running northwest and southeast through the site of Huancavelica. This fracture may mark the faulted crest of a great anticline. Near the city at any rate, the structure of the rocks is anticlinal. Cross fractures are indicated and Singewald [75] regards the two sandstones that he has mapped as opposite limbs of an anticline the axis of which strikes southwest and the nose of which has been faulted out.

Fig. 12 has been drawn from the maps and descriptions of Umlauff and Singewald to illustrate the ore occurrence at the Santa Barbara mine. The section runs southwest and northeast at right angles to the general strike of the ore deposits.

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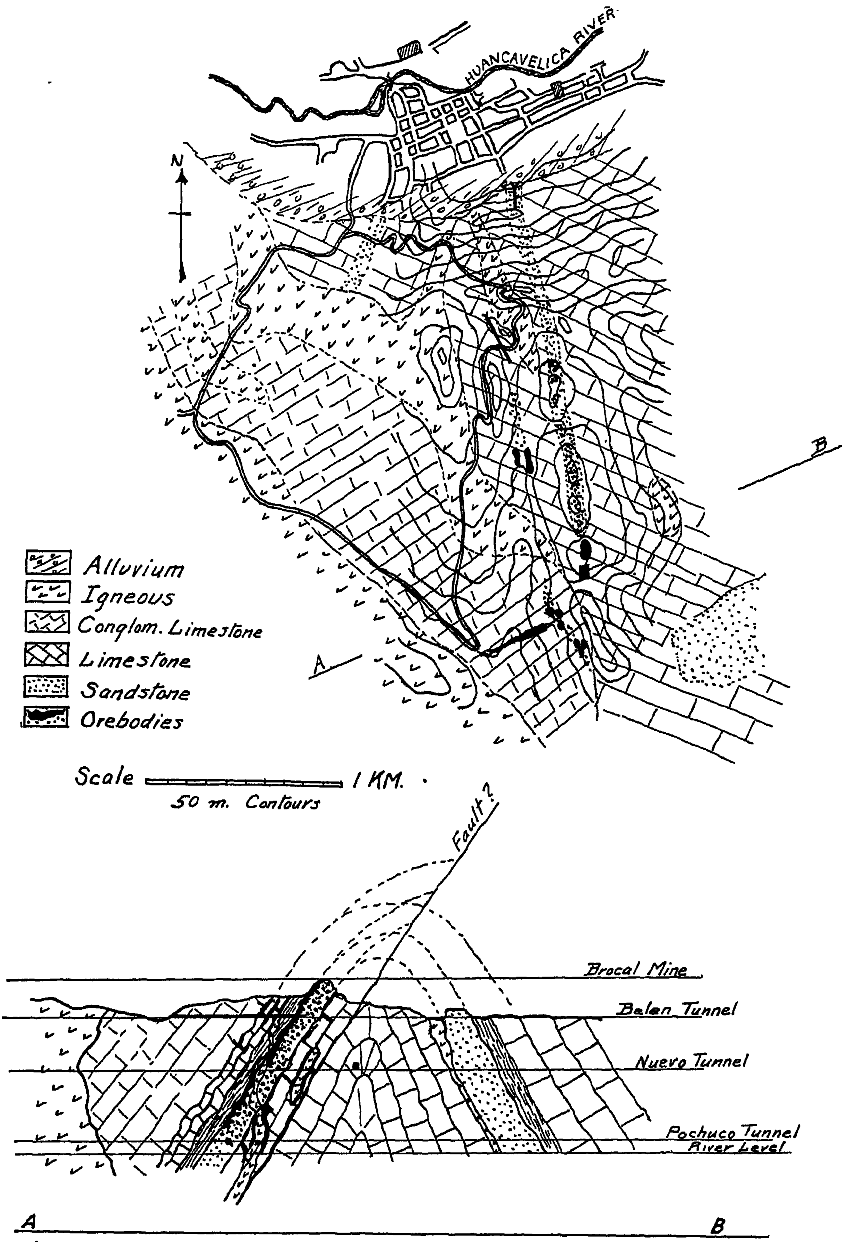
[73] G. F. Becker: *Op. cit.,* 20.

[74] A. F. Umlauff: El Cinabrio de Huancavelica. *Bol.* 7 del Cuerpo de Ingenieros de Minas de Peru (1904).

[75] J. T. Singewald, Jr.: The Huancavelica Mercury Deposits, Peru. *Eng. & Min. Jnl.* (1920) **110,** 518.



HUANCAVELICA

FIG. 12.—QUICKSILVER DEPOSITS IN PERU.

The principal orebodies were deposited in the sandstone bed, which is capped by a bed of shale. The rocks are greatly shattered and intruded by small tongues of andesite. The sandstone is characterized by many fractures across the bed, adding these voids to its inherent porosity. It therefore constituted an ideal receptacle rock in which the primary mineralization could be concentrated under the impervious shale capping.

The many faults, folds and igneous intrusions give evidence of the intense disturbance that the region has undergone. Hot springs give evidence of comparatively recent magmatic activity. The intrusives are premineral and the mineralizing solutions probably followed the intrusion of the andesite along the same fissure through which these dikes advanced to their present position.

The pregnant solutions circulated through the porous sandstone and through all other fissures and open rock spaces in the limestone, shale and conglomerate as well. The richest orebodies were formed in the sandstone where the solutions were trapped under the shale cap.

Umlauff states that the shales which cap the limestones and sandstones are continuous beds in which dislocations are rare. In the sandstone the cinnabar is found in stringers perpendicular to the stratification, in grains and nodules and in massive impregnations of the sandstone mass. The limestone contains cinnabar in fractures and joint planes, in lens-shaped masses and as cavity linings. The limestone conglomerate has cinnabar disseminated through the voids and it is also found in fissures in the conglomerate which had been loosely filled with detritus of both the limestone and basalt.

The Huancavelica quicksilver deposit, one of the largest of the world, is an excellent illustration of the theory of orebodies formed by primary concentration. Cinnabar-bearing solutions from probably the parent magma of the andesite intrusions rose through a major fissure or series of fissures made or followed by dikes of andesite over a distance of some 60 km. Orebodies were found only at localities where the solutions were trapped in voids of rock overlain by impervious cap rocks. Thus arrested and confined the pregnant solutions deposited their load of mineral matter in concentrated form to make the immensely rich orebodies mined by the Spaniards 300 years ago.

The district lies at an elevation of 14,000 to 15,000 ft. and is difficult of access. Labor and climatic conditions no doubt leave much to be desired. Despite these drawbacks the possibility of finding another structure favorable to the formation of large high-grade orebodies somewhere along the 60 odd kilometers of known mineralization should be an incentive to prospecting and exploration. Ten per cent. ore such as the Santa Barbara mine is reputed to have had is $300 ore at present quicksilver prices. That would make a good mine even when judged by the tenet that damned all mines that would not stand bad management.

Many other metals are mined by American companies in Peru and the commercial control so established is regarded as a distinct asset to the national independence of metal supply. With so essential a metal as quicksilver, production of quicksilver in Peru controlled by Americans would seem a venture which upon investigation would prove well worth attempting by our mining companies.

Quicksilver Deposits in Italy

Idria

Idria, formerly in Austria but in Italy since the war, has been the scene of quicksilver production for over 400 years. The mining town of Idria is 35 km. from the railroad station at Loitsch or 43 km. from St. Lucia, Tolmein, on another railroad. The elevation is 332 meters above sea level. The town lies in a deep valley at the confluence of the Idritza River and Nikova Creek. The mountains in which the valley lies are a southern extension of the Julian Alps. The valley of the Idritza River is steep-walled and runs in a roughly north and south direction. Many small tributaries of the Idritza have dissected the country, giving to it a rough broken appearance. A large northwest-southeast fault has dislocated the strata along the general strike of the formations.

The geology of this district has been intensively studied and extensively discussed and debated by many authorities. Fig. 13, which illustrates the ore occurrence, has been drawn from figures and descriptions of Lipold,[76] Kossmat[77] and Pilz.[78]

A notable feature in this district is the intense dislocation with both normal and reverse faults and the relatively small amount of folding that has affected the strata. The main northwest-southeast dislocation is accompanied by several parallel minor faults. Cross fractures can also be traced on the surface. The various faults of the main fracture system converge toward the northwest and southeast and just east of the town a north and south fault crosses them. At the crossing point the ground is greatly broken up. This zone of greatest fracturing marks the location of the ore deposits. The formations that have to do with the ore occurrence, and which are shown in Fig. 13, are:

1. Middle Carboniferous (Paleozoic) shales and sandstones. They are gray to black in color, carry pyrite and bituminous matter. The clay shales are dense and on cleavage planes they exhibit a micaceous glimmer.

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[76] M. V. Lipold: Erläuterungen zur geologischen Karte der Umgebung von Idria in Krain. *Jahrbuch* der K. K. Geolog. Reichsanstalt (1874) **24**, 425.

[77] F. Kossmat: Ueber die geologischen Verhältnisse des Bergbaues von Idria. *Idem* (1899) **49**, 259.

[78] A. Pilz: Zinnobervorkommen von Idria unter Berücksichtigung neuerer Aufschlüsse. *Glückauf* (1915) **51**, Pt. 2, 1057, 1081, 1105.

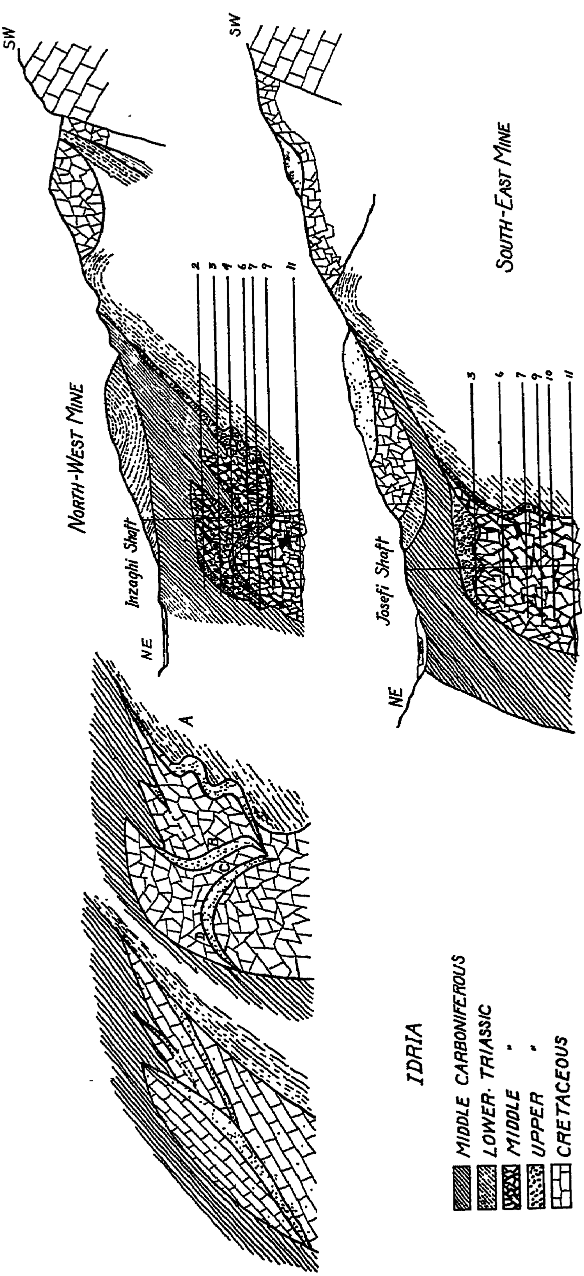

FIG. 13.—IDRIA MINE, ITALY.

2. Lower Triassic sandstones and sandy calcareous or dolomitic shales carrying mica. They are light in color.

3. Middle Triassic calcareous shales and interbedded limestones (Kossmat includes these with the Lower Triassic), dolomites and dolomite breccias.

4. Upper Triassic limestone and dolomite conglomerates, tuff with chert inclusions and marly shales, limestones and calcareous shales.

5. Cretaceous limestones 300 meters thick are faulted against the other formations on the southwest.

North West Mine

The North West mine, worked through the Inzaghi, Theresia and Franz shafts and the 400-year-old Antonio adit, has been explored to a greater extent than the South East mine. The underground workings afford an opportunity to study the complicated tectonic relationships.

The orebodies are in the Middle Triassic brecciated rocks and are bounded on the north northeast as well as on the south southwest by overthrusts. These overthrust rocks converge so toward the surface that they almost meet at the level of the Antonio adit and give to the orebody a wedgeshaped terminus. The trend of the overthrusts is west northwest-east southeast.

The northern boundary of the orebody, called the north contact, is formed by the dense impervious Middle Carboniferous shales which cap the orebody. In the lower levels this contact has a steep dip which flattens out above the third level, as shown in Fig. 13*c*.

The shales that comprise the cap rock are older than the Triassic rocks under them and hence represent an overthrust. On the cap rock are Lower Triassic strata, which normally should lie under them. These, according to Lipold, probably slid down from the hill to the southwest after the deposition of the Cretaceous strata.

The Middle and Upper Triassic rocks, chiefly dolomites, which are the receptacle rocks in which the orebodies were formed, are brecciated. This brecciation is thought to be due to the overthrust. Kossmat assumes the thrust to have come from the northeast and pictures the action as an overthrust accompanied by a partial overturn, as illustrated in Figs. 13*a* and 13*b*. He holds that the Lower Triassic resting on the cap rocks were overthrust at this time also, as against Lipold's thought that they slid down from above at a later time.

The Upper Triassic shales and sandstones are very important in mining because they often contain high-grade ore and, being continuous, are used as guides in prospecting. Fig. 13*a* shows these beds at the beginning of the overthrust and 13*b* shows how they were torn into fragments and mixed in with the brecciated dolomite. These beds when

first found in the mine and before their true relations were disclosed were named *A*, *B*, *C* and *D* from southwest to northeast. *A* was the footwall bed; *B*, the center bed; *C*, the southwest limb of the curved bed and *D*, the northeast leg of the same bed.

Bed *A* is the most continuous both horizontally and vertically. It consists of black shales and sandstones. The latter sometimes contain inclusions of the dolomite breccia and often grade into it.

The other sheets are less regular. Bed *B* is folded into a Z shape, and *C* and *D* beds into an arch which reaches to the north contact. As shown in Figs. 13*b* and 13*c*, the four beds unite in depth. Besides these main beds several minor fragments of these shales and sandstones are found in the dolomite breccia, such as the one sketched in on the north contact and the one between *A* and *B*.

The Middle Triassic dolomites and dolomite breccias are divided into upper and lower parts by the *C-D* bed. The lower part consists of massive dolomites and breccias which contain only small and irregular inclusions of the shales.

The dolomites confined between beds *B*, *C*, *D* and the cap rock have more shale inclusions and near the larger orebodies the dolomites are breccias or conglomerates merging into sandstones. The rocks between *A* and *B* beds are also capped by the Carboniferous shales and here the breccia is coarse.

The deposition of ore followed the overthrust. The Middle Carboniferous shales everywhere form a definite boundary of mineralization.

South East Mine

The workings of this mine, reached through the Josefi and Ferdinandi shafts, are continuous with the northwest workings on the strike but have a simpler tectonic structure. Here also the Middle Carboniferous shales form the north contact and the cap of the orebody. The south contact shows similar characteristics as farther to the north. Of the Upper Triassic beds only bed *A* is found as a continuous sheet while the others are represented by fragmentary sheets only, of which a number are well exposed in No. 4 level.

The principal ore-bearing rocks in the southeast group, then, are the typical dolomites and dolomite breccias of the Middle Triassic. The character of the orebodies in these rocks is different from that in the bedded Upper Triassic beds. The cinnabar in the breccia is distributed netlike through the interstitial space of the breccia as against heavy impregnations in the bedded sandstones. The evidence of the overthrust is as clear here as in the northwest group. In the lower levels open fractures lined with cinnabar and with walls heavily impregnated with cinnabar are found. These are evidently the feed fissure of the orebody. Evidence of volcanism in the region is given by the tuffs, though the deposition of these in Upper Triassic times no doubt preceded the

ore deposition. The tectonic disturbances in which the Cretaceous strata played a part were probably allied to a recurrence of volcanic activity, and ore deposition followed these disturbances, so that the ore deposition was probably not earlier than the Eocene.

In this ore deposit, overthrust and overfolding brecciated Middle and Upper Triassic sandstones, shales, dolomites and limestones. At the same time this movement covered the receptacle rock so formed with a cap of impervious shales. Fortunately, the underlying magma to which the fractures penetrated contained quicksilver which, escaping from the magma, was caught and concentrated in this most splendid trap which nature had provided for its reception and retention.

The ore is cinnabar, richest in the open-textured sandstones, rich in the dolomite breccia according to the degree of brecciation and leanest in the bedded dolomites, where only the bedding planes and cross fractures provided room for deposition. Native mercury occurs along the hanging wall near the contact with the Carboniferous shales carrying bituminous matter.

Metacinnabar has been found but according to Schrauf[79] it was far younger than the cinnabar and had formed since the opening of the mines.

Monte Amiata District

The history of the Italian quicksilver mines in Tuscany is similar to that of other quicksilver districts. They were worked in the early Greco-Roman periods for the cinnabar, which was used as a pigment. In the Siele and Morone mines Etruscan weapons have been found. Ancient stopes of the Cornacchino mine yielded tools, vessels and coins, one of the latter being a gold coin of Philip of Macedonia. In the twelfth and thirteenth centuries at least four of the mines had a period of production until war and pestilence forced a shutdown.

Beginning in the middle of the nineteenth century renewed attention was given to these deposits in increasing measure and the modern history of the mines dates from that time. In view of the occasional agitation in this country against our extralateral right law, it may be of interest to note that the development of these mines was greatly hindered by the ancient Etruscan laws which gave to the owner of the surface the mineral thereunder. The owners overvalued their land and it was difficult to combine a reasonably large tract from the many separately owned parcels. The owners themselves, lacking capital, experience in mining, and restricted to their small individual parcels of land, could not achieve economic production. One company, it is reported, needed three years' time to acquire a larger tract for mining purposes and had to make 300 separate purchases to do so.

[79] A. Schrauf: Uber Metacinnabarit von Idria und dessen Paragenesis. *Jahrbuch* der K. K. Geolog. Reichsanstalt (1891) **41**, 366.

The region and the quicksilver deposits in it are interesting geologically and have been described by Rosenlecher,[80] Spirek,[81] Kloos,[82] Lotti,[83] Muller,[84] De Castro[85] and others. Their descriptions have been used in preparing the following paragraphs on the ore occurrence and the illustrations of Fig. 14.

The quicksilver deposits of Tuscany lie along a major fracture extending some 25 km. from Monte Amiata in the north to Montebuono in the south. Monte Amiata, an extinct volcano, dominates the country with its elevation of 1734 meters. Cross fractures occur at many points along the course of the major fissure and near these crossings the most important orebodies are found. Sulfur and gas springs are numerous in the region, particularly near the Solfarate mine.

Fig. 14*a* illustrates the distribution of the orebodies along the major fracture. Fig. 14*b* shows a section parallel to the cross fissure at Monte Amiata.

The rock formations of the district that have to do with the ore occurrence are trachyte; a large lava flow from the craters of Monte Amiata covering the slopes of the mountain and surrounding country to the extent of about 100 sq. km. in all. This trachyte has the usual porous texture and gray color. In the mine, circulating waters have decomposed it at many places to a sandy mass. That it is ideal for storing water is attested by many excellent springs on the mountain slopes which foster luxurious vegetation in an otherwise dry country.

The youngest sedimentary beds that have to do with the orebodies are yellow-gray quartz sands with a calcareous or argillaceous binder, which may or may not cement the aggregate. Calcite and quartz veinlets abound in this rock, which varies from a predominantly clayey mass to one having the appearance of a fine-grained limestone. These beds, which are thought by some geologists to be Miocene and by others to belong to the upper Eocene, lie flat and unconformably on the beds below them. The orebody of Montebuono, to be described later, lies in these sands.

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[80] R. Rosenlecher: Die Quecksilbergruben Toscanas. *Ztsch. f. prak. Geol.* (1894) **2**, 337.

[81] V. Spirek: Das Zinnobererzvorkommen am Monte Amiata. *Ztsch. f. prak. Geol.* (1897) **5**, 369.

Das Zinnobervorkommen am Monte Amiata, Toskana. *Idem* (1902) 297.

[82] H. Kloos: Zinnober führende Trachyttuffe vom Monte Amiata im südlichen Toscana. *Ztsch. f. prak. Geol.* (1898) **6**, 158.

[83] B. Lotti: Die Zinnober und Antimon führenden Lagerstatten Toscanas und ihre Beziehungen zu den quartären Eruptivgesteinen. *Ztsch. f. prak. Geol.* (1901) **9**, 41.

[84] H. E. Muller: Der Quecksilberbergban in Toskana. *Glückauf* (1912) **48**, No. 6, 218.

[85] C. De Castro: Le Miniere di Mercurio del Monte Amiata. Memorie Descrittive della Carta Geologica d'Italia (1914) **16**.

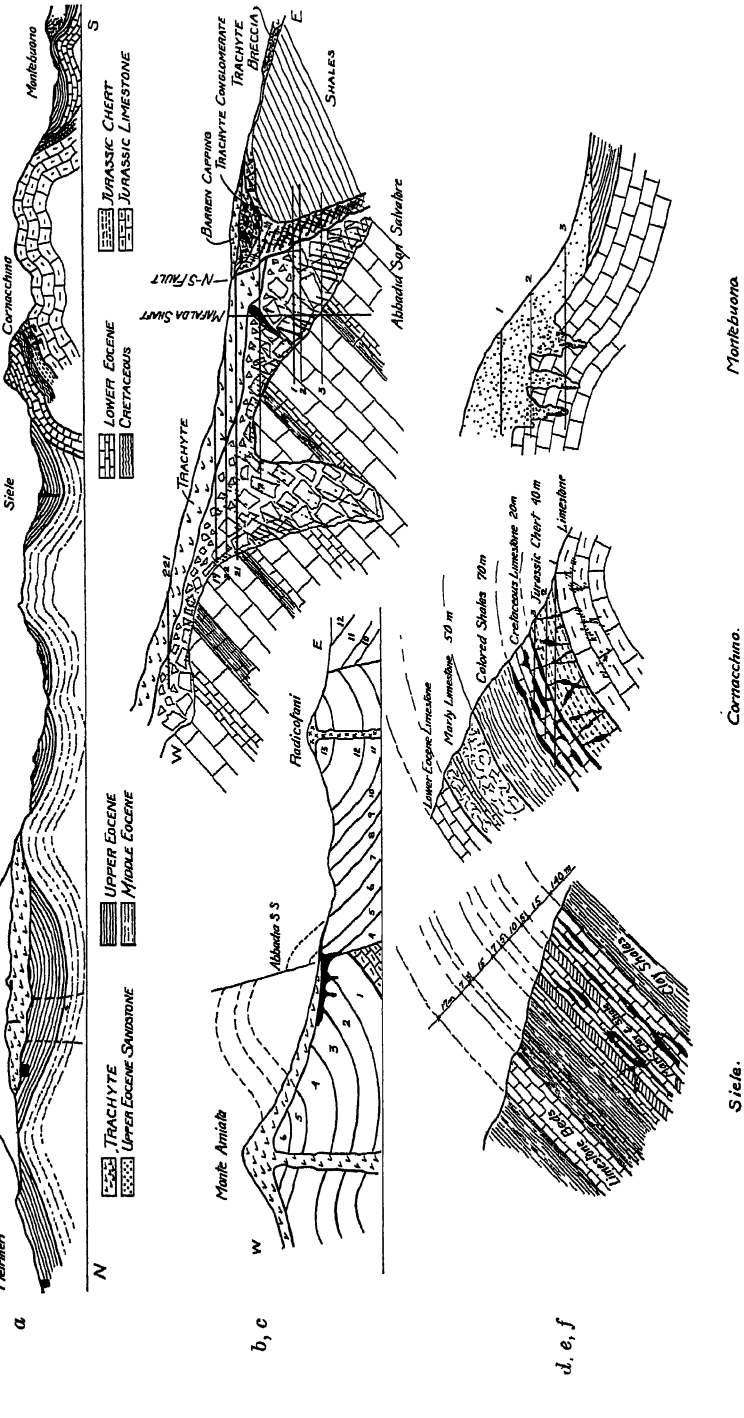

FIG. 14.—VARIOUS QUICKSILVER MINES IN ITALY.

The distribution of the Eocene strata is indicated in Fig. 14*a*. Three general divisions can be distinguished. The upper Eocene consists mainly of sandy to marly shales, gray to dark gray-green clay shales, and limestones intimately interstratified. The transition to the middle division is gradual and here gray to green-gray shales, reddish limestones and light gray homogeneous limestones in thin beds predominate. In the upper and middle divisions serpentine is found, on the contacts of which with the Eocene strata cinnabar has been found. The lower Eocene strata are typical Nummulitic limestones, which form the peaks of Monte Penna, Monte Nebbiaio and Monte Civitella.

Most of the quicksilver orebodies lie in the Eocene strata. Near the orebodies the limestones are interstratified with clay shales, which are generally much altered and may be wet or dry.

Conformably under the Nummulitic limestone is a zone of reddish marly shales interbedded with Nummulitic limestone and fine-grained homogeneous limestone beds containing streaks and nodules of a dark gray chert. This zone probably marks the transition from Eocene to Cretaceous. Contiguous to this zone and with the same dip to the north as exposed at the Cornacchino mine (Fig. 14*e*) are probably Upper Cretaceous reddish marly limestones some 50 m. thick. Under these are varicolored manganiferous clay shales 70 m. thick.

Under these clay shales, which form the impervious cap of the Cornacchino mine, lies a fractured, cavernous limestone 20 m. thick. It varies from hard limestone to marly decomposed material, contains chert inclusions and has solution chambers partly filled with clay and limestone fragments. These solution cavern breccias, similar to those of the Mexican and Texan deposits, are more or less impregnated with cinnabar at the Cornacchino mine.

This limestone bed is Lower Cretaceous in age and rests on a 40-m. bed of chert which is Jurassic in age. The chert bed is fractured and cinnabar has been deposited in the fractures.

Below the chert are gray, dense limestones which in places are completely decomposed to marly masses impregnated with cinnabar. These limestones are also Jurassic in age and represent the oldest rocks in which cinnabar has been found in this district.

The various ore deposits in the Monte Amiata district no doubt have a common origin from ascending magmatic waters. Their alignment along the major fracture system with the substantiating indications of solfataric springs all point to this conclusion. In this district, as in certain quicksilver districts of Nevada and Mexico, antimony in the form of stibnite is found, sometimes in the same orebodies with the cinnabar, indicating a common origin and similar mode of deposition for both.

The Siele Mine

The Siele is one of the oldest and at one time was the most important quicksilver mine of the district. It lies on the bank of a creek running in an easterly direction and apparently following the course of one of the cross fractures of the region. The ore horizon is a series of limestone beds interbedded with clay shales having an aggregate thickness of some 85 m., as illustrated in Fig. 14*d*. The footwall of this series is a bed of clay shales 140 m. thick. The limestone member of the ore-bearing series varies from light gray hard limestone to marly and clayey masses and often merges imperceptibly into the clay shales. Calcite veinlets traverse the limestone in all directions. The clay shales vary from shales to dry or wet plastic clays. The general strike is west northwest, east southeast and the dip about 45° north northeast.

Rosenlecher describes the orebodies as large flattened "pockets" perhaps 100 m. long along the strike and 100 m. wide, with a thickness of up to 2½ meters.

These pockets are solution chambers along the bedding planes partly filled with a mixture of clay, limestone fragments, marl and cinnabar. Ores containing up to 60 per cent. in quicksilver were found. These sheetlike orebodies pinch out in depth to a sometimes capillary thinness. The mine was abandoned as being worked out at least twice in its history because the respective orebodies pinched in this manner.

The footwall of the orebodies is generally solid limestone covered with a layer of calcite. The hanging wall is generally a marly limestone of a clay shale not sharply separated from the clay filling. Pyrite is an accompanying mineral, as is usual in quicksilver deposits. Pockets of high-grade cinnabar and clay mixtures are found in crevices in the footwall.

Clearly, these conditions are similar to those found in the quicksilver mines of Texas and Mexico. The same type of trap formed under impervious clay shales served to concentrate the primary mineralization and so, fortunately, caused the deposition of an orebody instead of allowing the escape and dissemination of the mineral-bearing solutions.

Solfarate Mine

This mine lies northwest of Siele and apparently in the same series of rocks but on another cross fissure. The mine takes its name from the numerous gas and sulfur springs in the vicinity. The geology and ore occurrence are essentially the same as that of Siele.

Cornacchino Mine

This mine lies on the south slope of Monte Penna. The ore occurs in Lower Cretaceous limestones and in Jurassic cherts and limestone beneath. The orebodies vary in character according to the physical

and chemical characteristics of the receptacle rocks but have a common origin. A section through the mine is given in Fig. 14*e*. Ore was discovered in the chert outcrop in 1879. In this rock the ore is found in small stringers generally running east and west through the brown brittle chert beds some 40 m. thick. The chert near the outcrop dips 15° to 20° north and the stringer fissures are generally perpendicular to the direction of the dip. Remarkably little pyrite is found with the cinnabar. This ore averaged 0.3 per cent. quicksilver.

By driving north through the chert in search of ore, the Lower Cretaceous limestone was reached. This was found to be 20 to 25 in. thick and in it quite unexpectedly bodies of ore were found that were rich beyond the wildest dreams of the operators.

The form of the limestone orebodies is extremely irregular and follows solution cavities. At first glance their arrangement seems equally irregular, but by mapping them, two main directional lines, following probably the original fracture planes, can be discerned. The chert limestone contact is wavy in section and exhibits a merging of one into the other. The limestone-manganiferous shale contact is also wavy, though it does not conform to the convolutions of the lower contact. The limestone-shale contact is sharp.

The limestone bed is now dry, thus testifying to the impervious character of the overlying shales in keeping out surface waters. It contains the usual solution caverns filled with marl and clay. These clay marls are impregnated with cinnabar and form the high-grade orebodies while the hard limestones contain cinnabar only in the crevices and fractures. The clay-marl orebodies are seldom greater than 2½ to 3 m. in vertical extent and 1½ to 2 m. in horizontal extent. The individual ore chambers are invariably connected by small, sometimes minute fissures through which the mineralizing solutions entered and circulated. Pyrite is abundant and often cinnabar has been deposited on masses of pyrite.

The ore chambers that are filled principally with clay are not so rich in cinnabar as those in which limestone fragments are intermingled with the clay. This same observation was made at the mines of Guadalcazar, Mexico, where two grades of cavern filling called *almendrilla* and *cuesco* were distinguished by the amount of contained clay, and in the Texas mines, where the *jaboncillo*, as the filling is called there, varied in quicksilver tenor with the clay content.

Where more limestone fragments are present, the texture is more open and many small individual traps form in an aggregate of limestone fragments surrounded by clay walls. The average content of the run of mine ore from these deposits was between 1 and 2 per cent. The richest ore was found against the impervious hanging wall, where the greatest concentration took place.

The description of these orebodies shows clearly that the cinnabar-bearing solutions must have risen through fractures in the brittle chert bed. They entered the limestone sheet from below, following joint planes and fractures. These openings were enlarged by solution, leaving the insoluble clay in the openings so formed. The clay shales above the limestone formed an impervious cap rock, which restricted the mineralizing solutions to the limestone and lower strata. The contained minerals of the magmatic solutions were concentrated by deposition in the restricted space of the solution chambers and smaller voids in the receptacle rocks. The chert orebodies are the cinnabar-filled feeder fissures for the limestone orebodies.

Since the solutions apparently came up through the chert, it seems logical to suppose that the solutions originally must have risen through the underlying Jurassic limestones also, and that they may have formed orebodies there.

This indeed is true though the ore deposits later found in the Jurassic limestones are not directly below those in the upper limestones but lie farther east along the course of the major cross fissures. The Jurassic limestone here is greatly altered and solution chambers of several hundred cubic meters in volume were found in which the clayey filling was impregnated with cinnabar, though the average content was low. This clearly illustrates the importance of an overlying impervious stratum to confine the solutions to a definite horizon. The upper limestone orebodies had such a limiting stratum and rich orebodies were formed under it. In forming the lower limestone orebodies the solutions passed on, up through the fissured and hence not impervious chert. Instead of being definitely arrested they were only partly checked by the restricted passageway through the chert and only a very moderate concentration of the primary mineralization took place.

Montebuono Mine

Montebuono lies some 3 km. south of the Cornacchino mine and is capped by Miocene or upper Eocene sandstones.

The mine lies on a branch of the Fiora River, in the valley north of Montebuono, which separates the latter from Monte Nebbiaio. The ore-bearing stratum consists of a sandstone loosely cemented by an argillaceous or mildly calcareous material. The color varies from yellow to dark gray, depending on the cementing substance. The average thickness of the stratum is 20 to 30 m. At the surface these sandstones show impregnations and stringers of cinnabar. In drifting on these leads the Nummulitic limestones were found to underlie the sandstones in irregular peaks, as shown in Fig. 14*f*. A large fissure running nearly north and south was also found. It is probably part of the major fault which extends north to Monte Amiata.

This fissure, up to 2 m. in width, has been opened for several hundred meters in length and over 40 m. in depth. It is also filled with the sandstone which here contains masses of clay and is impregnated with cinnabar. The sharp torn walls of the fissure indicate a violent rupture.

Near this fissure are several openings that become larger toward the surface in a funnel shape. These also are filled with clayey sand mass and are impregnated with cinnabar. The quicksilver content of these funnel-shaped masses increases with depth. The upper levels average 0.2 per cent., the middle levels 0.3 per cent. and the lower levels 0.4 to 0.5 per cent. The walls of these funnels are generally covered with a layer of crystalline calcite and are gashed by many small fissures. In these are found small bunches of ore up to 5 to 6 per cent. in content. The average run of mine ore is about 0.4 per cent. quicksilver.

Since both the main fault fissure and the accompanying funnel-shaped rifts show sharp angular walls and are filled with the same substances, it is reasonable to suppose that they have a common origin. Probably the funnel-shaped openings represent cross rifts in the limestones which are due to the anticlinal nature of the folding at this point. A rough jagged profile as described would be produced by such folding and fracturing of the crest of a limestone anticline. The main north and south fracture and these cross rifts were then filled from above by sand, which in time became more or less cemented.

The magmatic solutions rising from their source through the great north-south fracture spread through the porous sandstone mass filling the fissures and rose to the surface. Concentration of the primary mineralization occurred only in such small traps as were formed by little cross fractures in the walls of the large fissures. In the absence of any cap rock over the sandstone the pregnant solutions spread out through the sandstone and formed a low-grade deposit of cinnabar above the feed fissures. At the surface these magmatic waters no doubt were diluted by surface waters to the point of losing their identity. It is significant that the mineral content increases as the narrower parts of the fissure are reached in depth, as with depth dilution with surface waters would be progressively less. Here the gangue material was deposited in an open limestone fracture from above and the ore came in from below. In the Thirty-Eight mine in Texas both the ore and the gangue were deposited from above. Both mines testify to the long periods of time through which fissures in limestone can remain open.

Abbadia San Salratore Mine

This mine, now the most productive, was long considered the poorest prospect of the region. It lies on the east slope of Monte Amiata near the edge of the trachyte flow from this extinct volcano.

A strong east-west cross fracture is indicated and cinnabar indications are found for some 2½ km. along its course. In the early work at this mine cinnabar was found entirely within the trachyte blocks at the contact of limestone and trachyte and entirely within the limestone. The first named type of mineralization is a totally decomposed trachyte colored with cinnabar. It averages about 0.1 per cent. quicksilver.

On the trachyte-limestone contact a clay seam was found in the limestone, which dipped into the hill some 45°. It is indicated in Fig. 14*c* near the shaft. Under this clay seam high-grade ore was found and followed some 60 m. along the strike. Excessive water and running ground led to the abandonment of the workings. Tongues of the trachyte that penetrated the fractures in the limestone also showed clay on the contact and concentrations of cinnabar under them. These early attempts to explore the shattered Eocene limestones found occasional rich ore but were abandoned on account of trouble with water and running ground. Rosenlecher in 1894 mentions that there seemed little hope that the companies then prospecting the various mines would attain any notable measure of success.

By 1899 the area comprising the present mine workings had been consolidated and a new era of exploration and exploitation began. A section through the mine on the eastern slope of Monte Amiata is given in Fig. 14*c*. The east-west cross fault lies south of the section; *i. e.*, towards the observer. The area north of the cross fault dropped and a subsequent collapse of the escarpment south of the fault line is thought to have formed the breccia which through later cementation by a silicified kaolinitic ground mass formed the so-called "trachyte conglomerate." A subsequent flow of trachyte from the craters of Monte Amiata covered this trachyte conglomerate to a depth of 10 meters.

Below the mine, an area covering some 12 to 15 sq. km. is strewn with partly decomposed trachyte blocks with some marly clay filling between them. It probably represents a broken-up remnant of the trachyte lava sheet similar to the blocky remnants of the basalt sheet at the Oat Hill mine in California. This trachyte breccia is slightly impregnated with cinnabar but not sufficiently to class it as ore.

The trachyte conglomerate orebody was exploited beginning in 1899. The individual fragments are limestone, silicified shale, sandstone and black clay shale as well as trachyte blocks. The cinnabar is distributed irregularly between these fragments in the silicified clay ground mass. Some of the decomposed trachyte blocks show impregnations of cinnabar to a depth of 1 m. The cap rock of a subsequent trachyte flow covering this conglomerate was 10 m. deep, and barren. It was removed and the deposit was mined by open-cut methods. It was known as the "Lame" orebody. The average content of the "conglomerate" under the cap and down to a depth of 30 m. was 0.87 per cent. In depth

the ore tenor increased to over 1 per cent. This orebody has been followed down to over 100 m. below the surface in the brecciated zone of the two fault fissures.

The trachyte sheet covering the sides of Monte Amiata lies on the brecciated surface of the original slope. The breccia on this surface was formed probably by both tectonic movements and weathering action. It consisted of limestone, shale, clay shale and marl fragments from the strata comprising the original mountain slopes.

When the ore-bearing magmatic solutions, rising through the fault fissures, reached the trachyte, they were deflected and compelled to travel upward on the under side of it through the breccia of the former surface. In other words the fresh trachyte formed a relatively impervious cap over the coarse breccia through which the mineralizing solutions were forced to circulate. The rising thermal waters dissolved some of the limestone, adding to the voids for ore deposition and leaving the insoluble clay as an admixture of the ore. In time the interstitial space of the breccia became filled with a mixture of clay, cinnabar and sand from the decomposed trachyte. The ore bed itself now partook of the nature of an impervious rock mass.

At a later period, when the trachyte lava sheet had cracked and weathered sufficiently to give access to surface waters, these ran down hill under the trachyte on the impervious ore mass, which resulted in the decomposition of the under side of the trachyte flow into a mixture of sand, trachyte fragments and occasional fragments of the sedimentaries that had been picked up originally by the flowing lava. This explains Muller's statement that the ore forms some distance *under* the *solid* trachyte capping. Actually it formed against solid trachyte but this solid capping has been decomposed since deposition of the ore. Muller seems to have assumed that surficial waters diluted the mineralizing solutions but this is improbable, particularly when he clearly states that the heaviest mineralization occurs along the zone between limestone and trachyte fragments; *i. e.*, at the exact horizon where the original contact between breccia and solid trachyte must have been and where consequently one would expect to find the greatest concentration of cinnabar.

The ore mined from this mineralized zone under the trachyte capping varies from 0.5 to 1 per cent. A number of feeder fissures or funnel-shaped brecciated feeder zones have been found, as indicated in Fig. 14*c*. At least one of these extends downward into Jurassic strata, the position of which is indicated in Fig. 14*b*. These funnel-shaped oreshoots were formed probably about the intersection of cross faults.

Here at the Abbadia San Salvatore mine we have another excellent illustration of an orebody formed by primary concentration. At this mine the receptacle rock is of three origins. The Lame orebody was

formed in a breccia due to the collapse of a fault escarpment. The extension in depth of this same orebody is in the brecciated area of crossing faults and the hillside orebodies formed in the interstitial space of surface detritus. The cap rock, which compelled the concentrations of the mineralization in these receptacle rocks, was for all of them an igneous rock, as was also the case at the Mariscal mine in Texas and at the Oat Hill, Sulphur Bank and Knoxville mines in California. At Monte Amiata it was a surface flow of trachyte as against an intrusive sheet at the Mariscal but both served the same purpose, though in most quicksilver mines a fault gouge or a dense shale bed perform this service. In contrast to Sulphur Bank, California, the cinnabar here was not deposited in cracks of the covering lava flow.

At Monte Amiata, as at so many other large quicksilver mines, the large and rich deposits were unsuspected at first. Extensive exploration based on geological studies finally disclosed ore deposits of truly world-wide significance. We have in California a New Almaden and a New Idria which are worthy of their names. It would not be surprising if we were to discover a New Monte Amiata under the lava flows of Mount St. Helena near the intersection of Sonoma, Lake and Napa counties.

Quicksilver Deposits in Spain

There are three quicksilver-producing districts in Spain—Almaden, in central Spain, the Oviedo district south of the Bay of Biscay and a third group in the Provinces of Granada and Almeria in the southern part of the country. The most important district was and is that of Almaden, which lies 125 miles east of south from Madrid in the Ciudad Real district in the south central part of Spain.

This district is noted for three ore occurrences of major importance and many minor occurrences of cinnabar. The most important mine is that of Almaden, which has maintained its position as the premier quicksilver mine of the world since the inception of quicksilver mining, an astounding fact, having no parallel anywhere in other branches of metal mining. Another formerly important mine in this district is Las Cuevas, near Gargantiel, some 15 km. north of east from Almaden. The third large mine, now also abandoned, lies at Almadenejos, 10 km. south of Gargantiel.

In importance and interest Almaden so far outweighs all the other quicksilver mines of Spain that little has been published concerning them, and lacking a comprehensive description of their orebodies, only Almaden will be discussed here.

Fig. 15*a* gives a north and south section through Almaden. The region is one of Silurian and Devonian sedimentaries, greatly folded and faulted and intruded by igneous rocks. The section Fig. 15*a* was

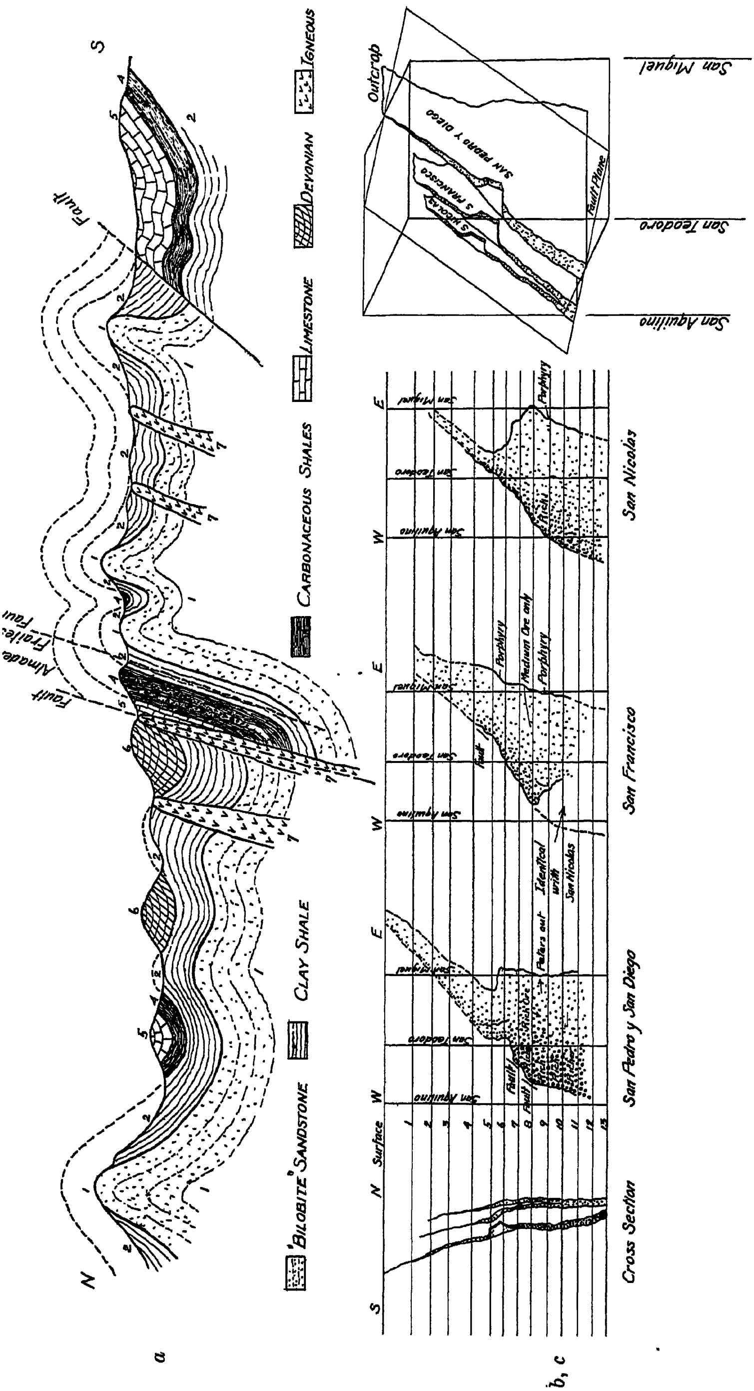

FIG. 15.—ALMADEN MINE, SPAIN.
a. North and south section.
b. Outline of orebodies as indicated by Kuss.
c. Block between the three shafts.

drawn after one given in "Minas de Almaden."[86] Besides this reference the writings of Bernaldez and Rua Figueroa,[87] Monasterioy Corre'a,[88] Kuss[89] De Kalb[90] and Ransome [91] were used in preparing the following paragraphs on the ore occurrence at Almaden.

The Silurian rock strata shown on the section are "Bilobite" sandstones, clay shales, carbonaceous shales with interbedded sandstones near the bottom and limestone. On these rest strata of the Lower Devonian. Diabase or other igneous intrusions are also indicated.

The ore deposit is found in a sandstone or quartzite interbedded with the carbonaceous shale series. On the footwall of the deposit is the much discussed *frailesca,* which has probably retained this distinctive name only because of the uncertainty of early investigators as to its true nature. On the evidence presented by Kuss and De Kalb it is assumed to be a fault breccia, or perhaps because of its fine texture it is better named fault gouge. The faulting which formed it occurred along the clay-shale stratum as indicated. The strike of the beds near the ore deposit is east and west and they stand nearly vertical. On both walls of the ore-bearing quartzite are strata of black slates or shales containing pyrite, which are bituminous in part.

Kuss gives a detailed description of the orebody illustrated by mine-level plans and a section. His description has been used in preparing Fig. 15*b*.

The cinnabar of the orebodies is contained in quartzite interbedded with black, impervious, carbonaceous shales. Three separate orebodies can be distinguished, separated by shales and quartzites. The walls of the orebodies are generally shales though in places dense barren quartzites and occasionally the frailesca gouge form the boundary of the orebody. At some horizons the ore will give out suddenly in one layer of quartzite and continue in an adjoining one.

Three shafts serve the mine. The main working shaft, called San Teodoro, is sunk in the footwall south of the deposit. San Aquilino shaft is in the western end of the mine and San Miguel shaft at the eastern end. The crosscuts from San Teodoro to the orebody first cut the frailesca gouge in passing from south to north. Kuss, referring probably to the area between the fifth and ninth levels, states that here it is firm

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86 Minas de Almaden. XIV Congreso Geologico Internacional, Madrid, 1926.

87 F. Bernaldez and R. Rua Figueroa: Memoria sobre las Minas de Almaden y Almadenejos (1864).

88 J. de Monasterio y Corréa: Mines de Mercure d'Almaden (Espagne). *Rev. Univ. des Mines* (1871) **29,** 1.

89 H. Kuss: Mémoire sur les Mines et Usines d'Almaden. *Ann. des Mines* [7] (1878) **13,** 39.

90 C. De Kalb: The Almaden Quicksilver Mine. *Econ. Geol.* (1921) **16,** 301.

91 F. L. Ransome: The Ore of the Almaden Mine. *Econ. Geol.* (1921) **16,** 313.

without great hardness and presents in a great many places drusal cavities lined with crystals of dolomite. Then come fractured shales and next the San Pedro y San Diego orebody in a white quartzite. This is the most important orebody and it is to be noted that it has the richest ore on its western end, the grade dropping off in depth (*i. e.*, vertically) and toward the east.

North of this orebody are alternating bands of quartzite and schist, then come the two orebodies of San Francisco and San Nicolas, separated from each other by a thin band of schists with bands of quartzite. These two formed in a black, harder and more compact quartzite; they are lower in grade than the San Pedro y Diego. The hanging wall is formed by black carbonaceous shales.

Bernaldez places the workings of the first level far to the east in the Castillo mine, which was worked through a crosscut tunnel. He says that it is impossible to decide of which orebody this may be a continuation, as the ore was present here in a ramified series of stringers which "spattered" the top of the quartzite crowning the hill above Almaden.

This agrees with Kuss' description of the trend of the orebody in the second, third and fourth levels. Fig. 15*b* shows the outline of the orebodies as indicated by this description. These figures are drawn in the plane of the orebodies, the projection of the three shafts being indicated. The faulting on the west end of levels 5, 7 and 8 is specifically mentioned. (Levels 9 and 10 had not been worked to the west end at that time.) Other evidence of faulting is furnished by the sharp curve to the south made by the three orebodies on the fifth and sixth levels. Other cross faults exist, as De Kalb mentions that the east end of the orebodies is cut off by a fault. A section through the three orebodies is given at the left of Fig. 15*b*.

Kuss in his level plans has sketched in the frailesca gouge near the Aqüilino and San Teodoro shafts. The latter has apparently been sunk in the gouge. From the San Aquilino shaft a crosscut runs through the gouge to the orebodies, which becomes shorter with each deeper level, and curiously enough the frailesca fault gouge at each level extends to the orebody. It seems probable then that an oblique fault which is marked by this fault gouge crosses the ore zone.

In Fig. 15*c* the outlined cube represents the block of ground between the three shafts. The oblique fault plane is indicated. The impervious gouge of this fault, sealing the nearly vertical sandstone beds on a 45° dip, was the means of directing and limiting the circulation of the mineralizing solutions to the area where the ore is now found. The greatest concentration of cinnabar should be found just under this fault gouge cap rock, and it is so found, the ore grading off from the cap rock outward. Instead of the cap rock being on the hanging wall as is usual, it lies across the strike of the orebody just as the mudrock at the Oceanic

mine in California formed a cap rock across the strike of the high-grade sandstone orebody of that mine.

The faulting, folding and the intrusives give ample evidence of a formerly active magma underlying this region. From Fig. 15*a* it can be seen that the ore-bearing strata of Carboniferous shales were cut by a fault and that this plane of weakness was followed by an intrusive dike. The mineralizing solutions no doubt followed the same path and thus gained access to the sandstone or quartzite interbedded with the shales. Whether the intrusive or branches thereof actually entered the ore-bearing strata, as for example the Elvan streak did at New Idria, has been hotly debated. Kuss reports a porphyry at the eastern end of the sixth and ninth level on the San Francisco orebody and on the ninth level of the San Nicolas. De Kalb denies the existence of igneous rock in the mine, though Ransome is inclined to credit it. The debate in the past generally concerned itself with the question: Is the frailesca a fault gouge or an altered intrusive? The frailesca no doubt is a fault gouge and very probably tongues of the intrusive penetrated it and later became altered almost beyond recognition, so that the two sides of the debate were both right and wrong.

Another much debated point on the Almaden orebodies is whether the cinnabar was deposited in preexisting pores of the sandstone or whether it replaced quartz. Becker, writing in 1887, states positively that the cinnabar does not replace quartz; Ransome, in 1921, is equally positive that it does. Ransome, in examining specimens from Almaden, finds that the cinnabar is not deposited exclusively in the interstices of a sandstone but that it has "replaced" both quartz and the interstitial sericite of the quartzite, entering the quartzite through small fissures.

The composition of the mineralizing solutions as they first entered the receptacle rock can only be guessed at, but it was no doubt undersaturated with respect to cinnabar and silica. At Almaden the process may have been as follows: A fault striking northwest and southeast cut obliquely across the east and west vertically dipping beds of sandstone. The fault plane was inclined and cut the sandstone beds on a 45° dip angle to the west. Alkaline thermal solutions from the magma below rose from the dike fissure through the cracks and fissures in the sandstone beds, until they encountered the impervious gouge in the inclined fault plane. Now they were deflected toward the east as they rose under the gouge and between the shale walls of the sandstone beds. As the solutions penetrated the sandstone through fissures, cracks and the int rstices of the sandstone they dissolved some of the silica, thus increasing the pore space of the sandstone. The dissolved silica was probably deposited farther up as the solutions became saturated with respect to silica. Since the greatest flow of the magmatic water would be just under the fault gouge, it follows that the greatest amount of leaching

would be done here. As the solutions rose they would approach the saturation point, both because of the increased load of dissolved silica and because of a probable drop in temperature and pressure. In other words, it is to be expected that the lower parts of the sandstone beds would show more leaching of the grains than the upper parts, if indeed the upper parts showed any leaching at all. Hence Becker examining ore from higher levels than Ransome 35 years later may have obtained differing results.

When deposition of cinnabar began it was deposited in all the open space available, hence the ore grows richer in depth as the leaching of the sandstone in depth here provided additional space in the receptacle rock for the deposition of ore; hence also the richest ore is found on the west just under the fault gouge where the greatest flow of solution took place. Increase of the grade of the ore in depth must be interpreted to mean at equal distances from the cap gouge, as vertical depth may mean a relative displacement eastward and consequent lower grade that would offset the gain in ore content due to depth.

The San Pedro y Diego sandstone was probably more porous to begin with than the other beds, as it seems to have been the main channel for the rising solutions. It may be that the white color of this sandstone is due to leaching or more probably the dark color of the San Francisco and San Nicolas beds is due to a different and more filling cementing material, which explains their lesser porosity and greater hardness. The occasional displacement of the ore from one layer of sandstone and its continuance in an adjoining one may be due to a cross fracture which allowed the rising solutions to cross a small impervious shale parting to a more porous adjoining bed of sandstone.

The close approach of the three orebodies in depth, making it probable that they will unite, points clearly to a common source of mineralization, as indicated in Fig. 15*a*. It seems probable that the sandstone beds thinned out near the top and that the wall shales capped them by closing over them. If not, hanging wall shales formed the seal, as an anticlinal fold was originally present at some distance above the present surface, as indicated.

It is obvious, then, that the greatest body of cinnabar ore thus far discovered upon earth was formed by conditions practically ideal for the concentration of the primary mineralization. The receptacle rock was a porous sandstone, the pore space of which was increased in depth by preliminary leaching. This receptacle rock was encased by walls of impervious shales. The top was practically sealed off by the same shales. Especially favorable conditions for concentration were provided by a fault gouge cutting across the receptacle rock at an angle of 45° with the horizontal, which directed and confined the mineralizing solutions to a stream along its underside.

Quicksilver Deposits in Russia

There was one important orebody of quicksilver in European Russia near Nikitovka, in the Donetz coal basin. This has been described by Tschernyschew and Lutugin.[92] A section through the mine is given in Fig. 16*a*. The age of the enclosing rocks is the Carboniferous.

At the mine a series of flat-lying sedimentary strata dip sharply. Fracturing occurred and is marked by a brecciated zone dipping steeply in the opposite direction, which cuts through the sedimentaries as shown. Under a cap rock of dense shale or slate there lie in descending orde a sandstone, a dense quartzite, another sandstone and another quartzite. The ore was found in the sandstone underlying the cap shale. Weber and Markow[93] say that the cinnabar is confined to the sandstone and is not found in the contact shales, as is sometimes reported. Coal seams are present in these strata and at least one report has it that cinnabar has been found with coal at one point in the mine. This compares with the occurrence of cinnabar with coal in the Barnum McDonnell mine in the State of Washington. Apparently the ore-bearing solutions rose through the brecciated fault zone and escaping into the porous sandstone stratum were confined to it by the shale capping and caused to concentrate into an orebody.

This mine was found in 1879 and reached a maximum annual output of over 18,000 flasks in 1897. Continuous operation ceased in 1911, though some production has been reported since then.

Quicksilver Deposits in China

There are a number of localities in China where cinnabar has been found but according to Tegengren[94] the important deposits are confined to Kueichow province and the adjacent parts of the neighboring provinces, Szechuan and Hunan in the northeast and Yunnan in the southwest.

The Kueichow plateau is a horst of gently folded limestones and shales from probably pre-Cambrian to Permian in age. The folding with a north-northeast south-southwest trend is post-Carboniferous while the great faults surrounding the horst are believed to be Tertiary. Tegengren's map of the region shows the quicksilver occurrences extending over a broad zone 100 km. wide running northeast and southwest for a distance of some 700 km. A characteristic of the northern deposits is that they occur in elevated anticlines. An even more general characteristic with regard to the territorial distribution of the deposits is that

[92] T. Tschernyschew and L. Lutugin: Guide des Excursions du VII Congrès Géologique XVI (1897) 36–45.

[93] W. Weber and K. Markow: Abstract, *Neues Jahrbuch f. Min.* (1927) 1, 33.

[94] F. R. Tegengren: The Quicksilver Deposits of China. Geol. Survey of China *Bull.* (1920).

they occur along several roughly parallel zones, coinciding at the same time with axis of folding and with the chief lines of dislocation.

The cinnabar of the mines is found in brecciated rock and in cracks and fissures of the rock, generally limestone interbedded with clay shales.

a

b

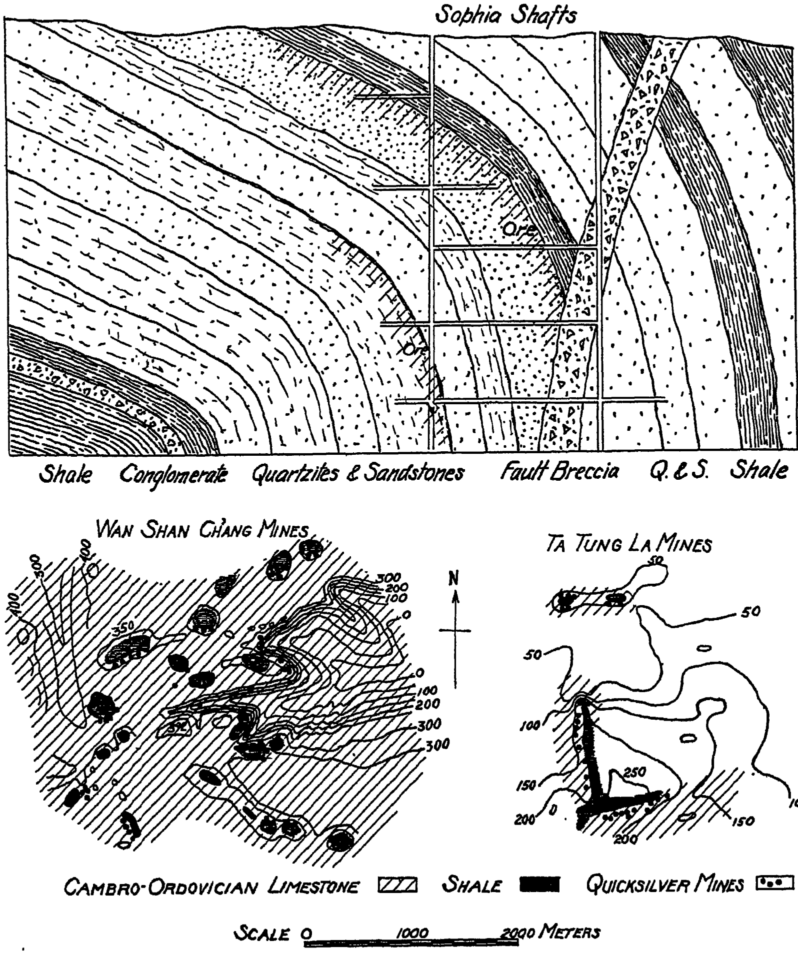

FIG. 16.—QUICKSILVER OREBODIES IN RUSSIA AND CHINA.
a. Section through quicksilver mine in Donetz coal basin, Russia.
b. Area near Wan Shan Chang mines, China.

Plate 3 of Tegengren's paper shows a map of the area near the Wan Shan Chang mines, at present the most important mines of the country. This map, reproduced in Fig. 16*b*, shows clearly that the quicksilver orebodies lie in the limestone just under the shale capping that overlies the ore-bearing horizon. Tegengren calls the cap rock simply a shale

but as he shows a photograph of a pottery works which utilizes the shales overlying the ore horizon, it seems clear that the cap rock can be classed as an impervious clay shale similar to the Del Rio clay on the Edwards limestone of the Texas quicksilver district.

The picked ore from the mines runs from 1.7 to 4.4 per cent. quicksilver. Mining in this region has been carried on since the fourteenth century, though in other parts of China quicksilver mining and reduction dates from at least 300 or 200 B.C. The ore is mined through tunnels driven into the hills just under the clay-shale stratum. Mining and metallurgical methods are primitive and no large-scale exploration is carried out.

Apparently the Chinese quicksilver deposits lie in a huge block uplifted above the surrounding basins by a tremendous tectonic disturbance in perhaps Tertiary time. Intrusions of igneous rock, probably from an underlying magma, penetrated the strata of this block. These intrusions followed old fractures or lines of weakness along the anticlinal folds. Quicksilver-bearing thermal solutions, rising along these same fissures as a later phase of the intrusions, were blocked at various horizons by shale strata, particularly in the anticlinal folds.

Concentrations of the primary mineralization took place at these horizons in any brecciated rock or in other open space in the rocks, and thus orebodies were formed.

In these deposits again, the importance or necessity of a structure favoring primary concentration is illustrated, as all the larger deposits described by Tegengren in this entire region possess the characteristics required by the theory of the formation of orebodies by primary concentration.

Conclusions

Many other quicksilver mines the world over could be cited to illustrate the theory of the formation of quicksilver orebodies by a primary concentration from the ore solutions. Demaret-Mons,[95] for example, describes the Vallalta-Sagron deposit in Venetia as being deposited by thermal waters in sandstone enclosed by an impervious coat or envelope. Where this envelope is missing the cinnabar has been disseminated throughout the mass of the rock.

The examples cited should suffice to show that, given a region in which quicksilver occurs, the orebodies should be sought where conditions favoring a concentration of magmatic solutions exist. When viewed and judged from this standpoint the frantic efforts to class the orebodies as veins, stockworks, beds, impregnations or what not are seen to be unnecessary. The question of whether or not they will extend to great

[95] L. Demaret-Mons: Les principaux gisements de mercurie du monde. *Ann. de mines de Belgique* (1904) **9**, 46.

depth is generally irrelevant, as only in exceptional cases does this materially affect the *extent* of the orebodies. A flat-lying orebody is preferable to one of similar extent dipping steeply and thus attaining depth, because the flat-lying orebody can be extracted through less expensive shafts than a vertical one. There is no inherent virtue in a quicksilver orebody merely because it goes to great depth. The age of the associated rocks and gangue minerals, of which Phillips[96] has compiled an interesting list, is more or less accidental. They are attendant but not causal occurrences.

Examples of quicksilver orebodies in many different parts of the world have been cited. At some of these, as at Montebuono in Italy, Opalite in Oregon and the B. and B. in Nevada, the mineralizing solutions reached the surface. Only a partial concentration of the primary mineralization took place and only low-grade orebodies were formed. Such surface-formed orebodies are generally marginal, depending on the price of the metal. Other examples, such as the Black Butte, Non Pareil and Bonanza in Oregon, the Castle Peak in Nevada, and the Arizona mines, formed low-grade orebodies only because of the tightness of the receptacle rock.

The largest and most productive mines, which had high-grade ore in quantity, are those in which the most favorable conditions for concentration existed. Fissures from the parent magma to the point of deposition exist. Impervious rock strata directed and limited the magmatic solutions to pervious rock masses. These receptacle rocks were breccias or sandstones of generous interstitial space. The ore mineral is the primary cinnabar precipitated in concentrated condition on account of loss of pressure and temperature of the primary alkaline thermal solutions. The shape of the orebody is determined by the disposition of the confining rocks.

These considerations are the important ones that determine not only the very existence of the orebody but also its size, shape, location and grade of the contained ore.

The latter factors in turn affect the prospecting, development, mining, metallurgy and economics of the quicksilver-producing industry, therefore a thorough understanding of the geology is of fundamental importance in quicksilver mining.

DISCUSSION

(R. J. Colony presiding)

W. Forstner, San Francisco, Calif. (written discussion).—The development of the theory of genesis of quicksilver ore deposits as outlined by Mr. Schuette is certainly of great value to those interested in the development of such orebodies, and is highly

[96] W. B. Phillips: Geology of Quicksilver Deposits. *Min. World* (July 25, 1908) 131.

appreciated by them. The limiting and even damming of ascending mineral-bearing solutions by relatively impervious rocks forming bodies of ore deposits is not limited to quicksilver ore deposits but is also found in deposits of other minerals.

Mr. Schuette does not mention in his paper the Socrates mine in Sonoma County, California. This deposit is interesting because of its relatively large content of native mercury. It is in the solfataric area on the west slope of Mount Cobb. The main ore deposit lies at the contact of a belt of serpentine, forming the top of a ridge, and an underlying belt of sandstone forming the slope of the ridge. The grade of this contact fluctuates but may be considered as averaging from 45° to 50° NW. For a certain distance from the contact, 10 or 20 ft., the serpentine has been highly silicified and indurated, while the underlying sandstone is very friable close to the contact, hardening to some extent further away from the serpentine. The ore deposition was a later phase of the silicification process, the contact of the serpentine and the sandstone being the main channel system along which the ore deposition took place, while the silicified and indurated serpentine formed the capping. The native mercury occurs mainly as minute globules in the more compact portions of the rocks. This would suggest that the solfataric emanations reached the present ore zone, at least partly, in a gaseous state, and where the mercury vapors entered through capillary fractures into the hard rocks and cooled there suddenly they formed native mercury, while in the wider channels, where cooling occurred gradually, the formation of cinnabar by alkaline waters took its regular course.

Again an indorsement of Mr. Schuette's statement that "No two orebodies are exactly alike in their geological relationships."

C. De Kalb, Tucson, Ariz.—When I had the privilege of visiting the world-famous quicksilver deposit of Almaden in July, 1919, I was fully posted in advance upon the literature pertaining to it. It was, therefore, a great surprise when the sub-director, Senor Gonzalo del Rio, in charge at the time, said that some geologists who had visited Almaden had been in error concerning the occurrence of igneous rocks directly associated with the ores. Placed thus on my guard, it was evident that particular caution was necessary in taking my specimens of rocks. In fact, I even had Senor del Rio and one of the mine foremen take samples of suspected rocks in addition to my own. I also took specimens from various levels to make doubly sure. Underground many of these specimens certainly had the appearance of being of igneous origin, and possessed it on being taken outside the mine. I felt certain that my Spanish friends must have been mistaken. Especially did the apparent dike following the north wall of the San Nicolas lode seem, on megascopic examination, to possess marks of being a porphyry. My surprise was great, therefore, when the result of Dr. Ransome's microscopic investigation, based upon the samples, was to confirm the statement of Senor del Rio. It is certain that underground the sampling was thoroughly done. I am quite ready to accept the suggestion of Mr. Schuette that the *frailesca* no doubt is a fault gouge and very probably tongues of intrusive penetrated it and later became altered almost beyond recognition.

It may be of interest to add that small quantities of mercury occur widely distributed throughout parts of Arizona. I am familiar with cinnabar and metallic mercury derived apparently from mercurial tetrahedrite in the Roadside mine, in the Coyote Mountains of southern Arizona. The ore is essentially copper-bearing, with considerable silver. The tetrahedrite is an intimate intergrowth with bornite and chalcocite, usually occurring in injected seams. Pyrite is almost wholly absent, but small quantities of chalcopyrite in seams may have contributed to the formation of cinnabar in alteration. The cavities accompanying the cinnabar are plentifully filled with remnants of the original injected seams of bornite-chalcocite-tetrahedrite. Samples from the upper 200 ft. of the ore often show 2 to 3 per cent of mercury, and a

general average of the upper portion of the mine is 0.25 per cent. Other small deposits of mercury are found along the extension of the Baboquivari-Coyote range in southern Arizona.

C. N. Schuette (written discussion).—Mr. De Kalb and Mr. Forstner both mention the occurrence of native quicksilver. Mr. Forstner's mention of the Socrates mine brings to mind the fact that native quicksilver seems to be characteristic of several mines in that vicinity. The Esperanza mine, the Thompson and Baumeister prospect, and the Rattlesnake mine are other examples. Whether condensation from gas played a part in this deposition is perhaps an open question. I am not inclined to believe it, as conditions never exclude the possibility that the native quicksilver may be a product of secondary alteration. In fact, conditions where it is found, such as nearness to surface, organic matter in the rock and associated minerals, generally favor this supposition.

I had occasion to visit the Thompson and Baumeister discovery ½ mile above the Esperanza mine when it was but a week old and there found specimens of what is undoubtedly montroydite occurring as a secondary mineral with the native quicksilver. In the upper levels of the Red Elephant mine in Lake County, native quicksilver is found in the rock; there also in association with secondary montroydite.

Mr. De Kalb's mention of mercurial tetrahedrite in Arizona suggests other rare quicksilver minerals. The Skaggs Springs mine in Sonoma County, now in the development stage at shallow depth, shows no cinnabar in parts of the orebody. On panning, a tail of black material, probably metacinnabar, is found. Apparently this is not in sufficient amount to account for the assay value of the panned material. This ore has a green to yellow tinge, and while part of this color is due to Curtisite, there may be Terlinguaite or Eglestonite in the ore. In other parts of the mine the ore mineral is cinnabar practically to the surface.

I am glad to note that Mr. De Kalb accepts my suggestion that tongues of intrusive probably penetrated the frailesca, even though he found no intrusive directly associated with the ore. The association of quicksilver orebodies with igneous rocks, generally intrusive dikes, is so marked that an unqualified exception of the importance of the Almaden orebody would be startling.

Occurrence of Petroleum in North America

By Sidney Powers,* Tulsa, Okla.

(New York Meeting, February, 1931)

Contents

The purpose of this brief account of the occurrence of petroleum in North America is to make available to those who have a knowledge of geology, but who are not engaged in the petroleum industry, a few salient facts about the location of oil and gas fields, their productivity, the important structural types of accumulation and the age of the rocks containing petroleum. A few useful data and statistics are furnished, yet not complete statistics that are available elsewhere. Geology has been treated in simple terms because the reader is presumed to have little interest in the names of formations and of oil sands.

Mineral deposits are more or less analogous. Whether oil or metals, minerals are found in rocks, largely in sedimentary rocks which carry water or have at one time carried it. A genetic or accumulative

* Amerada Petroleum Corporation.

relationship between water, gas and minerals is common; hence the story of petroleum is useful to those engaged in other branches of the mining industry.

Distribution of Fields

The oil and gas fields of North America are distributed widely in rocks of many different ages associated with diverse geologic conditions (Fig. 1). In the East the fields follow the trend of the Appalachian Mountains. Gas occurs in the highly folded rocks and farther west both oil and gas in the rocks that are more gently folded. Still farther west, in an area of gently undulating folding, both oil and gas are found in the east-central oil fields of Ontario, Ohio, Indiana, Illinois, Kentucky and Michigan. West of the Mississippi River the fields of Kansas, Oklahoma, northern and western Texas and southeastern New Mexico are grouped in the gently tilted and folded Mid-Continent region, bounded on the east by the Ozark uplift, on the west by the Rocky Mountains and almost bisected by the Arbuckle and Wichita mountains in southern Oklahoma.

Prolific oil and gas fields have been found in the very gently folded rocks of the Gulf Coastal Plain (the Cretaceous and Tertiary embayment) south of the Mid-Continent region, the producing horizons being of Cretaceous and Tertiary ages. Along the Gulf Coast itself, oil, but little gas, is found associated with salt domes and in a few uplifts beneath which salt has not yet been penetrated.

The Rocky Mountain oil fields are mostly on pronounced anticlines in the folded area within the Cordillera and east of the true Front Range of the Rockies. They extend from New Mexico to the Turner Valley field near Calgary, Alberta, Canada. The fields are widely scattered and, with the exception of Salt Creek, Turner Valley and Kevin-Sunburst, do not cover large areas. Production is largely from rocks of Cretaceous age except in the last two fields, the first of which produces from limestones of Mississippian age and the second from the unconformable upper surface of limestones of the same formation.

On the Pacific Coast the oil fields are confined to the southern half of California near the Coast Range, in areas of both steep and very gentle folding. The San Joaquin Valley, Los Angeles Basin and Santa Barbara coast have been the productive areas. The oil comes from the Tertiary. There is comparatively little gas in the shallow, heavy oil fields, but large volumes in the deep, lighter oil fields of the Los Angeles Basin and in the Kettleman Hills field of the San Joaquin Valley.

Canada has one important new oil field, Turner Valley, near Calgary, Alberta, located about 20 miles east of the Rocky Mountains but within the area of mountain movements. A number of small gas and heavy oil fields have been developed in slightly folded rocks underlying the plains farther east. The old fields in Ontario are approaching exhaustion.

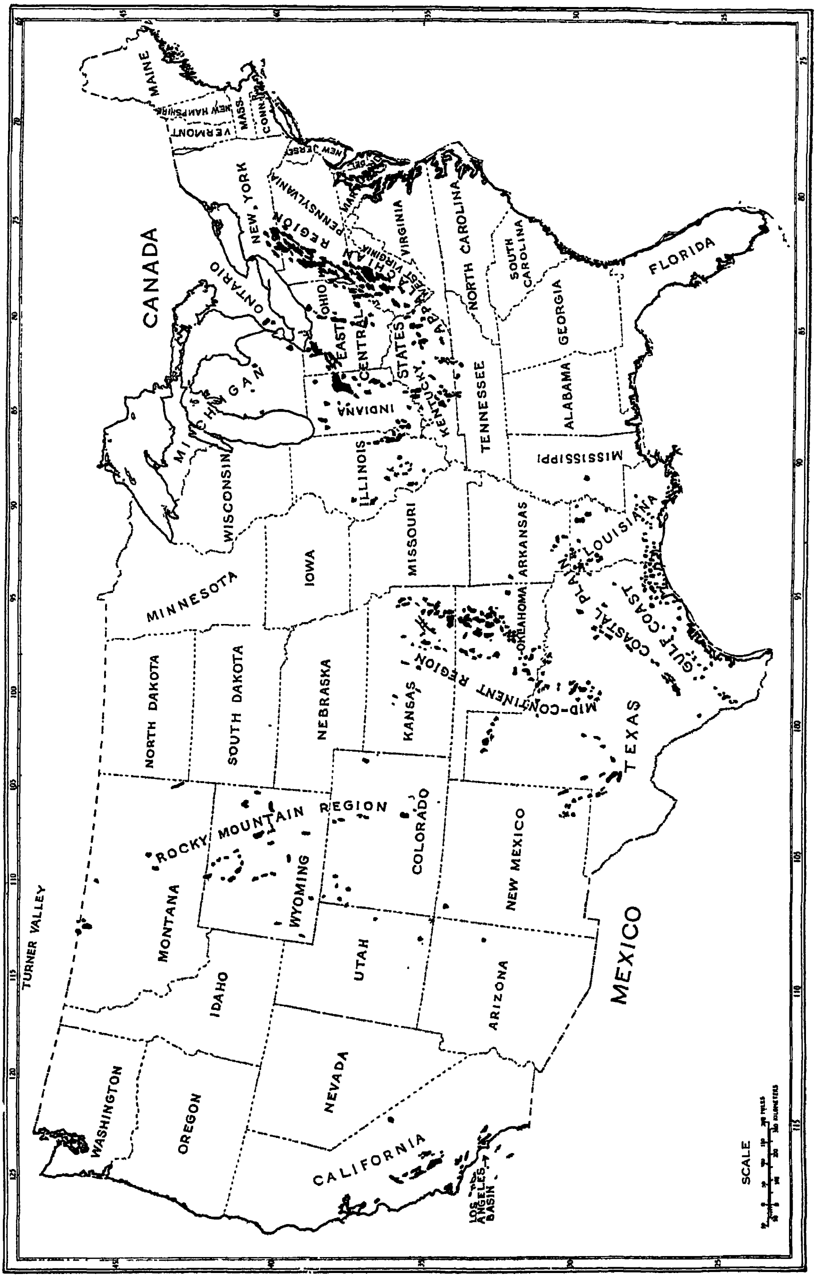

FIG. 1.—LOCATION OF THE OIL AND GAS FIELDS IN THE UNITED STATES AND CANADA.

In Mexico (Fig. 2) the oil fields are confined to gently folded rocks of the Atlantic Coastal Plain; the northern (Panuco) fields west of Tampico, southern fields south of Tampico, and the less important salt-dome fields of the Isthmus of Tehuantepec. The oil in the fields near Tampico comes from limestones of Lower Cretaceous age, that in the fields of the Isthmus from sands of Tertiary age.

All of the production east of the southern Rocky Mountains and north of the Coastal Plain is from strata of Paleozoic age. Moreover, oil has

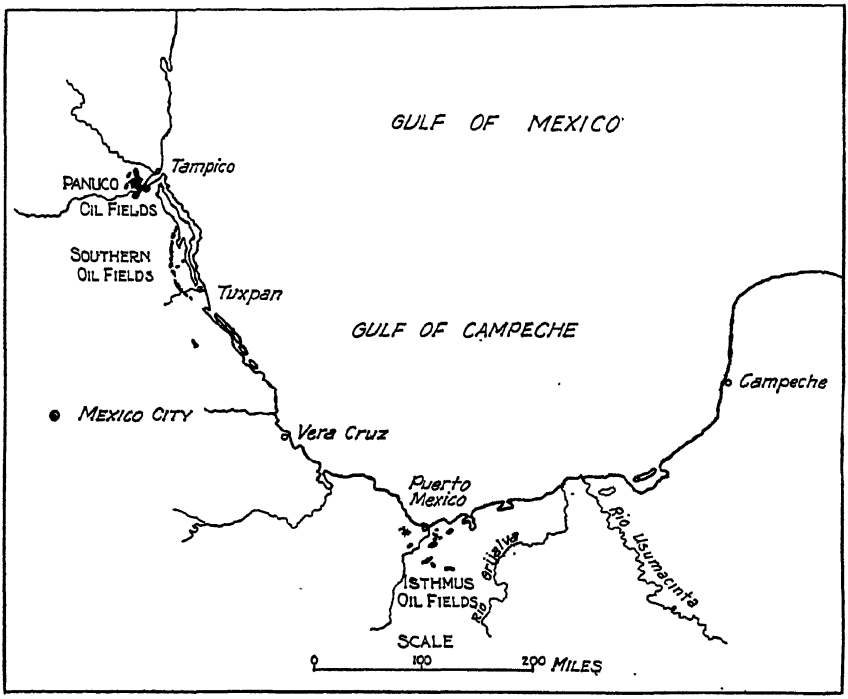

FIG. 2.—LOCATION OF THE OIL FIELDS IN THE TAMPICO-ISTHMUS OF TEHUANTEPEC AREA, MEXICO.

never been found in commercial quantities in Paleozoic rocks elsewhere in the world except in Bolivia, Argentine and near Perm in Russia. A. Beebe Thompson has estimated that 41 per cent. of the total production for the world has come from Paleozoic rocks, 15 per cent. from Mesozoic rocks and 44 per cent. from Tertiary rocks.

HISTORY OF DEVELOPMENT

Oil was first shown to the white men in 1627 by the Seneca Indians in the western part of New York State. The occurrence of "fluid inflammable substance" on the coast of Santa Barbara, Calif., has been known since 1792. Wells drilled for water in Pennsylvania as early as 1829 flowed oil, but commercial development is dated from 1859, when Colonel

Drake found oil in western Pennsylvania at a depth of 69 ft. Exploitation of oil and gas fields has kept pace with the mechanical development upon which modern civilization rests. Oil was first used as a medicine. As soon as the elements of refining were discovered, kerosene replaced whale oil as an illuminant. Natural gas came into use for the same purpose. Not until the internal combustion engine was perfected was there a demand for gasoline (petrol) and fuel oil. During the past decade the automobile has brought about the modern methods of discovering, recovering and refining oil.

Seepages of oil and gas and outcrops of asphalt led to drilling for oil in the Coastal Plain and Gulf Coast. The Lucas gusher at Spindletop, Texas, in 1901, located because of seepages, brought about the drilling of salt domes along the Gulf Coast and also on the Isthmus of Tehuantepec where there were similar seepages. Oil was discovered in the Tampico region because of drilling at very large seepages.

California and Rocky Mountain oil fields were discovered as a result of drilling at seepages, asphalt outcrops and asphalt lakes. In 1864 400 bbl. of oil were recovered from an oil spring on the Ojai ranch, near Santa Barbara, California.

During the past 20 years geology, with the recent addition of geophysics, has been the accepted indicator of new oil fields.

Origin of Oil[1]

Opinions differ as to the origin of petroleum, and several theories have their followings. The problem of the genesis of the oil itself is the most fundamental and important of the many problems of petroleum geology. Most American geologists believe that oil is derived from organic matter deposited in sediments as slimes (sapropels), oozes, or water-plant debris, together with animal matter, in relatively tranquil or stagnant water and now lithified into shales or limestones. The vestiges of plant remains recognized under the microscope consist mainly of algal filaments, one-celled micro-algae, coverings of pollen grains, spore envelopes, coloring matter from plant and animal cells, fragments of woody cell walls, chitinous debris, wax and resin fragments, etc. Of these, algae appear the most important. Animal products resulting from smothered decomposition of the organic debris probably enter into the organic colloids in the sedimentary deposit. Diatom cases are abundant in shales associated with certain oil deposits, especially in California. It is believed by most geologists that the deposition of source rocks of oil took place in saline waters, but this is not definitely proved.

Dynamic and chemical metamorphism in the course of geologic time, which itself seems to be an important factor, caused dehydration and progressive elimination of the volatile material from the organic sedi-

[1] David White, of the U. S. Geological Survey, has kindly revised this section.

ments, concomitant with gradual lithification. The ultimate results of the alteration are gases and liquid hydrocarbons and progressively carbonized residues. David White has shown that commercial deposits of normal oil are not found in regions where the fixed carbon (pure coal basis) of the humic coals in oil-bearing or overlying, younger formations exceeds a limit of about 61 per cent; and that beyond this point gas, asphalt residues and unusually high Baumé gravity oils are found in small amounts only. He believes that the production of hydrocarbons takes place in nature at a temperature of less than 225° F.

Experimental tests have shown that oils artificially distilled from fresh-water oozes do not differ notably either in volume or quality from those obtained from marine deposits; hence salinity may not be a prerequisite to oil generation.

The common source rocks of oil are believed to be shales and limestones containing organic matter of sedimentary origin.

Structure, Accumulation and Migration

The occurrence of oil and gas in North America has been found to validate the structural theory of the late I. C. White that, under uniform conditions of porosity and in the presence of gas, oil, and water, gas occurs up the dip and water down the dip from the oil. When the structure of the petroleum-bearing horizon is considered (and not the structure at the surface, which may be quite different) it is found that the important oil fields of North America are all on anticlines. The exceptions are explained by reservoir conditions (variations in porosity or sand thickness or texture), absence of connate water, or local structural features (such as faults, unconformities, salt-dome intrusions, abrupt changes in rate of dip, etc.).

Oil and gas fields in North America have been found in areas of gentle folding. The structure is simple. It is true that the folds of the Appalachian geosyncline have dips sufficiently steep to be plainly visible, but they are simple.

Exceptions are found in California, where a few small fields have been opened on the flanks of steeply folded mountain ranges. The notable exception is Turner Valley, Alberta, Canada, where the structure is a long, sharply folded, "napped" (overthrust) anticline cut by thrust faults. Both folding and faulting were caused by the pressure of a huge overthrust; the Turner Valley area was formerly just above the sole of the overthrust Front Range of the Rocky Mountains, now eroded back a distance of 20 miles. Drilling through this sole elsewhere has located oil in rocks of Cretaceous age under rocks of pre-Cambrian age.

Simplicity in the structure of North American oil fields stands in sharp contrast with complexity in the structure of many European oil fields which are within complexly folded mountain chains.

Oil and gas move in porous reservoirs, a fact proved by the behavior of old wells when new wells are drilled near by and also by the movement of water and oil in gas drive and water-flooding, and of chemical substances put in one well which come out in another. The rate of movement depends directly on porosity of the reservoir rocks. The commonly accepted view in the United States is that oil and gas do not move far from their original source to the reservoir from which they are produced; also, the migration is believed to be lateral or vertical; upward, not downward.

Reservoir Rocks

The common reservoir rocks are sandstone ("sands"), limestone, dolomite, chert and arkose. Serpentine acts as a reservoir rock in Cuba, serpentine and tuff in a few fields in south-central Texas, and basalt and baked marls at the igneous contact in the Furbero field of Mexico (now exhausted). Uplifts caused by basaltic intrusions do not form reservoirs of commercial importance in Mexico, published sections notwithstanding, but some of the dikes which cut the anticlines have increased the porosity near by and in that way have concentrated accumulation. The largest yields of oil from single wells have come from cavernous limestone in Mexico and Persia.

Production from limestone and dolomite is obtained, as a rule, in the upper, porous surface of thick limestone beds. This fact has led to the speculation that such surfaces were once exposed to subaerial or subaqueous erosion with accompanying development of porosity. Exceptions to this rule are found in the Lima-Indiana fields, where oil occurs several hundred feet within the dolomite as well as near the upper surface, in Turner Valley, at Oklahoma City and in the fields of the Texas Panhandle and Howard County, Texas.

The porosity of oil sands varies within such wide ranges that average figures are difficult to give. The average for each entire producing sand body is lower in the Appalachian fields than in other fields, according to A. F. Melcher, and is highest in California and Gulf Coast fields. The range of porosity in oil sands is from about 7 to 40 per cent.

The average diameter of sand grains in the oil sands of New York State is 0.15 mm., in the "Wilcox" sand of Oklahoma from 0.1 to 0.2 mm. The size of grain is larger in the arkose of the Amarillo fields, Texas, in certain central Texas fields and in some of the sands of the Appalachian district. The diameter in these exceptional instances is 1 cm. or more, but the mean diameter seldom exceeds 0.3 millimeter.

The capacity of sand with an average porosity of 20 per cent. is 8700 cu. ft., or 1550 bbl. per acre-foot, or 207 metric tons of 35° Bé. gravity (0.850 sp. gr.) oil. Such a sand 50 ft. thick can hold (but not yield) 77,500 bbl. per acre, or 10,000 metric tons of this grade of oil.

Gas is present almost everywhere in solution in light oils. The average content in Oklahoma is 2000 cu.ft. of gas per barrel of oil. Gas produced with the oil is called casinghead gas and usually contains sufficient gasoline to permit commercial recovery of this product.

Dry gas (free from oil) is obtained from both porous and well-cemented sands of low porosity. In the Appalachian fields, where the reservoir sands are cemented, wells having an initial volume of less than 1,500,000 cu. ft. a day are of commercial value because of long life, and are productive under vacuum for many years. In the Mid-Continent region, however, wells with an initial volume five times as large are not commercial, because they have a short life, due to small reservoirs, greater porosity, and trouble with bottom and edge water. In California gas is forced underground and stored in exhausted gas and oil sands.

The percentage of recovery of oil from reservoir rocks is determined by studies of chunks and cores. It is thought that on the average 15 to 20 per cent. is recovered by present methods. In the Bradford field, Pennsylvania, the sand has an average thickness of 38 ft., porosity of 16 per cent. of the entire volume of the rock, and yields 8 per cent. of the oil by pumping methods. An additional 20 per cent. will be recovered by water drive and still more by the use of soda ash and other devices. The Bartlesville (Pennsylvanian) and "Wilcox" (Ordovician) sands of Oklahoma, according to A. F. Melcher, have a porosity of about 10 to 35 per cent.; the average recovery, by present methods, from the former has been 10 to 30 and from the latter 15 to 60 per cent. Unconsolidated fine sands in California and on the Gulf Coast range in porosity from 25 to 40 per cent. and 10 to 30 per cent. of the oil is ultimately recovered.

Mining of oil sands has been attempted experimentally in Texas, Colorado, and California.

Methods of Drilling and Exploration

Most of the United States and Canada is sectionized; that is, divided into sections of one square mile, or 640 acres. This facilitates orderly development of oil fields. The common spacing of wells in the Mid-Continent fields is 300 or 330 ft. from property lines, which is equivalent to one well to 10 acres. Where property lines are irregular or production is shallow or limited in extent, or where there are several producing horizons, the wells are more closely spaced. In some of the Gulf Coast and California oil fields the derricks are within a few feet of each other.

Only two methods of drilling are used in North America, the percussion ("standard" or "cable tools") and the rotary. In California and on the Coastal Plain and Gulf Coast, almost all wells are drilled with a rotary. In the Mid-Continent fields, many deep wells are drilled nearly to the oil sand by the latter method and, after casing is set, are drilled

into the sand with standard tools. All wells are produced through flow lines connected at the casing head and are not permitted to flow freely into the derrick. Vacuum has been used for many years to aid recovery in old fields and gas-lift (forcing gas down into the oil column to make the well flow) is now used in new fields. Water-flooding is used at Bradford, Pennsylvania.

The problem of finding and of extending oil fields in North America, besides deeper drilling for new oil-bearing horizons, is essentially a search for obscure or buried structure; buried mountain ridges, like the Nemaha Mountains (Nebraska-Kansas-Oklahoma); buried uplifts, like the Bend Arch of Texas; buried anticlines, like Big Lake, Texas; buried hills like Healdton, Oklahoma; and buried salt domes. With rare exceptions, such as Burbank, Okla., the problem is to find anticlines. Anticlines of limited area underlain by oil-bearing strata (as, Eldorado, Kansas, in contrast with barren anticlines farther north) and with suitable reservoir conditions (as the fields along the buried Amarillo Mountains) must be sought on these large buried structures.

Where exposures are adequate and the structure at the surface is similar to that underground, even though it may not be parallel to that on the oil-bearing horizon, surface geology is mapped where the dips are steep by observed dips and strikes; where the dips are gentle, by elevations on key beds located and determined by planetable and telescopic alidade. Airplane maps are an excellent means of locating formation contacts, lines of outcrop and faults, and such maps provide an accurate base for detailed work.

Subsurface geology, the correlation of well logs, is now a most valuable means of predicting the discovery of oil by drilling wells before they reach the oil sand, of indicating proximity of anticlines or of uplifts on which there may be small anticlines, and of delimiting producing fields. Correlations depend for their value on the accuracy of the data and the spacing of the wells. Cuttings and cores from most new wells in North America are now saved and studied for micropaleontologic and micropetrologic content. The development of oil fields is dependent on correlations of formations made from data obtained in this manner.

In areas of unsatisfactory exposures, core drilling, in most areas with diamond drills, is the most satisfactory method of obtaining correlations and measuring dips and locating faults.

When core drilling is not feasible because of structural discordance or of excessive cost, geophysical methods are used. The seismograph has been very successful in locating salt domes and anticlines. The torsion balance is also of recognized value in locating salt domes and, in certain areas, other oil-bearing structures. Pendulums are used for the same purpose. Magnetic observations have delimited certain pronounced, buried areas of unusual magnetic anomoly, but have not

proved satisfactory in locating anticlines, faults and salt domes of small areal extent. Other geophysical methods are being tried.

Oil-field Statistics

Table 1 shows the relatively small area of the United States underlain by developed oil fields. Table 2 gives statistics regarding oil in Mexico, where most of the production has come from near Tampico, and Table 3 gives statistics for Canada. Canadian production declined steadily for a number of years, until the discovery of the Turner Valley field, Alberta. Table 4 shows occurrence of oil in North America, tabulated by age and type of structure.

Table 1.—*Oil-field Statistics, United States*

	Acres
Area underlain by sedimentary rocks	1,105,000,000
Area underlain by igneous or metamorphic rocks	827,000,000
Area productive of oil and gas in 1925	2,140,000
	Barrels
Total production through 1929	12,246,000,000
Total production in 1928	898,000,000
Total production in 1929	1,004,000,000
Average initial production oil wells completed during 1928	668
Average initial production wells drilled in 1929	396
Average production per well March, 1929	8
Average production per acre to end of 1929 for 34 largest fields	13,770
Light gravity oil (24° Bé., 0.91 sp. gr.) and lighter, produced in 1929	904,562,000
	Number of Wells
Total wells drilled for oil and gas through 1928, including abandoned wells and 157,000 dry holes	763,095
Producing oil wells, March, 1929	316,073
Of these wells, 6024 yielded one-half of the total production and had an average daily production of 232 bbl. each.	
Wells drilled in 1929	26,356
Of these 15,572, or 59 per cent., found oil; 2870, or 11 per cent., found gas; and 7914, or 30 per cent., were dry.	
Average gravity of oil from 34 largest fields, which have produced 6,902,000,000 bbl. through 1929, 30.8° Bé., 0.870 specific gravity.	

Appalachian Region

The Appalachian region comprises the oil and gas fields of New York, Pennsylvania, West Virginia and eastern Ohio. With few exceptions the fields are many years old, with the peak of production during 1891–1900. The wells are comparatively shallow (1000 to 2500 ft. deep) and they now

TABLE 2.—*Oil in Mexico*

	Barrels
Total production through 1929	1,585,000,000
Total for 1928, including the Isthmus	50,150,000
Total for 1929, including the Isthmus	45,000,000

	Barrels		Number of Wells
Northern fields (Panuco district) (12° Bé., 0.985 sp. gr.)			
Total production through 1929	637,000,000	Total number of producing wells drilled (594 producing Apr. 1, 1929)	1,478
Average daily production, Apr. 1, 1929	60,000	Producing area, acres	250,000
Southern fields (Dos Bocas-Alamo) (21° Bé., 0.927 sp. gr.)			
Total production through 1929	931,000,000	Total number of producing wells drilled (218 producing Apr. 1, 1929)	518
Average daily production, Apr. 1, 1929	42,000	Producing area, acres	70,000
Isthmus of Tehuantepec fields (32° Bé., 0.864 sp. gr.)			
Total production through 1929	17,000,000		
Average daily production for 1929	23,000		

TABLE 3.—*Oil in Canada*

	Barrels		Number of Wells
Total production through 1929	24,847,676	Oil wells in active operation end 1928 (40 at Turner Valley, Dec. 1929)	2,699
Production in 1928	624,184	Gas wells in active operation end 1928	2,037
Production in 1929 (952,300 bbls. Turner Valley)	1,120,693		

produce only a fraction of a barrel of oil per day. In 1928 only 4700 wells were drilled in this region (including 560 unproductive wells and many used for water-flooding) in contrast to 133,000 producing oil wells and probably over 50,000 gas wells. The density of the oil varies from 38° to 47° Bé., 0.833 to 0.791 sp. gr., and it has a paraffin base.

Oil and gas have been found in 56 individual horizons, mostly sands, in the Appalachian province. Only two sands are productive over most of the area, the Big Injun and Berea, of Mississippian age. Many of the sands, especially in the Devonian, disappear westward away from the former shore of the geosyncline. The average thickness of the productive part of oil sands is 25 feet.

Accumulation is controlled by (1) sand thickness and porosity, (2) water content within the sands and (3) structure, but reservoir conditions are more important than structure. Gas occurs along the crests of major anticlinal folds; in subsidiary anticlines on both major anticlinal and synclinal folds; and at the up-dip (overlapped) edge of sands. Oil is found down the dip from gas; in local anticlines; in sand lenses of adequate porosity (in part along old shore lines); and in local synclines within

TABLE 4.—*Occurrence of Petroleum in North America*

Oil Region and State	Predominant Type of Structure of Oil-bearing Rock in Order of Importance	Geologic Age of Producing Horizons in Order of Importance of Total Production	Lithology and Continuity of Oil-bearing and Gas-bearing Rock
		UNITED STATES	
Appalachian Region			
New York	Synclines, anticlines, terraces	Devonian (oil, gas) Silurian (gas) Ordovician (gas)	Lenticular sands (no water)
Pennsylvania	Anticlines. Long anticlinal ridges (gas on top); long synclines (oil in bottom or on flanks)	Devonian Mississippian (Pennsylvanian)	Sheet sands and sand lenses; also shoestrings (no water) Distribution controlled by sand conditions and local structure where water is absent
West Virginia	Synclines, terraces, anticlines	Mississippian Devonian Pennsylvanian	Sheet sands in general with occasional shoestring pools or lenticular sand areas (little water) (Miss.) Sheet sands and lenticular sands (Dev.) Lenticular and sheet sands (Penn.)
Eastern Ohio	Homoclines, anticlinal noses, terraces, overlaps	Mississippian Silurian (gas, oil) Pennsylvanian	Lenticular sands (Penn) Sheet sands (Miss) Sheet sand locally lenticular gas at edge of overlap (no water) (Sil.)
East Central States			
Western Ohio	Monoclines, anticlines, terraces, faults(?), buried hills(?)	Ordovician	Dolomitic part of Trenton limestone
Indiana	Anticlines, terraces, unconformities (?)	Ordovician Mississippian Devonian Pennsylvanian	Porous dolomite (Ord.); lenticular and sheet sands (Miss.) Lenticular sands (Penn.) Porous limestone (Dev)
Michigan	Anticlines	Devonian Mississippian	Porous limestone (Dev.) Sheet sand (Miss.)
Kentucky	Anticlines, faulted sand lenses, faults, synclines and terraces	Siluro-Devonian Pennsylvanian Mississippian	Dolomitic limestones (Sil-Dev.) Lenticular and sheet sands (Miss., Penn.) Shale (Dev. gas)
Tennessee	Homoclines	Ordovician Mississippian	Dolomite, limestone
Illinois	Anticlines, unconformities	Mississippian Pennsylvanian Ordovician	Sand lenses (Penn.) Sheet sands and lenses (Miss.) Dolomite (Ord)
Mid-Continent Region			
Missouri	Homoclines	Pennsylvanian	Lenticular sands
Kansas	Anticlines, homoclines unconformities, buried hills	Pennsylvanian Ordovician Mississippian	Shoestring and lenticular sands (Penn.) Porous siliceous limestone (Ord.) (Miss.)
Northern Oklahoma	Domes, anticlines, homoclines, faulted anticlines, terraces, unconformities, buried hills	Pennsylvanian Ordovician Devonian Mississippian Permian	Lenticular and sheet sands (Penn.) (Miss.) Sheet sands and siliceous limestones (Ord.) Porous limestone (Dev.) Lenticular sands (Permian)
Southern Oklahoma	Anticlines, terraces, buried hills, unconformities	Pennsylvanian Permian Ordovician	Lenticular sand (Penn., Permian) Limestone and sand (Ord.)
North and north central Texas	Anticlines, homoclines, terraces, faults, buried hills, unconformities	Pennsylvanian Permian Ordovician	Lenticular sands (Penn., Permian) Porous limestones (Penn., Ord.)
Western Texas	Anticlines, unconformities, reefs	Permian Pennsylvanian Ordovician	Porous limestone and dolomite (partly algal reefs) Lenticular sands Sheet sand (Ord.)
Southeastern New Mexico	Anticlines, terraces, reefs	Permian	Lenticular sands, porous limestones and dolomites (partly algal reefs)
Interior Coastal Plain			
Eastern, south central and southwestern Texas.	Faults, anticlines, homoclines, salt domes, volcanic tuff cones	Upper Cretaceous Eocene Lower Cretaceous	Widespread sands; limestone; sand and limestone domed over volcanic cones (now serpentine), serpentine.
Northern Louisiana	Anticlines, faults, unconformities, faulted anticlines	Upper Cretaceous Lower Cretaceous Eocene	Widespread sands (Up. Cret.), Chalk (Up Cret), Porous oolitic limestone (L. Cret), Lenticular sand (Eocene) Sand unconformity (Up.-L. Cret)
Southern Arkansas	Anticlines, faults, faulted anticlines	Upper Cretaceous	Lenticular and sheet sands
Mississippi	Anticlines	Upper Cretaceous (gas) Pennsylvanian (gas)	Sheet sands
Gulf Coast			
Southern Texas Southern Louisiana	Salt domes, faults, anticlines	Miocene Oligocene Pliocene Eocene Cretaceous (?)	Lenticular sands, porous cap rock (limestone)
Rocky Mountain Region			
Montana	Anticlines, domes, unconformities	Mississippian Upper and Lower Cretaceous Jurassic Pennsylvanian	Porous, eroded limestone at unconformity (Miss.-Jurassic contact) Sheet sand (Cret , Jura., Penn)
Wyoming	Anticlines, domes	Cretaceous Pennsylvanian Permian Mississippian Jurassic Ordovician Tertiary (gas)	Sheet sands, lenticular sandy shale (Up Cret , Penn., Jura., Ord) Jointed shale (Up. Cret.) Limestone (Miss.)

TABLE 4.—(*Continued*)

Oil Region and State	Predominant Type of Structure of Oil-bearing Rock in Order of Importance	Geologic Age of Producing Horizons in Order of Importance of Total Production	Lithology and Continuity of Oil-bearing and Gas-bearing Rock	Date of First Important Oil Production
		UNITED STATES		
Colorado	Domes, anticlines, homoclines	Upper Cretaceous Jurassic	Sheet sands, joint planes in shale; shale	1887
Northwestern New Mexico	Domes	Tertiary (gas) Upper Cretaceous Pennsylvanian	Sheet sands (Up. Cret.); limestone (Penn.)	1921
Utah	Salt anticlines, synclines	Pennsylvanian	Sands	(1908)
South Dakota	Anticlines	Pennsylvanian	Sandstone	1930
California	Anticlines, homoclines, faulted anticlines, faults, overlaps, buried hill	Pliocene Miocene Oligocene Eocene Upper Cretaceous	Sands, some lenticular; fractured chert. Part of the oil has accumulated against brea deposits	1875
		ALASKA		
Katalla	Steep folding	Tertiary	Fissured shale	1902
		CANADA		
Ontario	Anticlines, homoclines	Devonian Silurian Ordovician Cambrian?	Porous limestone and dolomite; sheet sand (not important)	1861
Alberta	Sharply folded and faulted anticlines (light oil), gently folded anticlines and overlaps (gas, heavy oil)	Mississippian Lower Cretaceous Jurassic Upper Cretaceous	Porous limestone and dolomite (Miss.); sands (part lenticular)	1914
		MEXICO		
Panuco	Fracture zones	Lower Cretaceous (Tamaulipas)	Fractured, porous limestone	1904
South Fields	Buried ridges, anticlines, faults	Lower Cretaceous (El Abra)	Cavernous limestone, lenticular sands	1908
Isthmus of Tehuantepec	Salt domes	Miocene, Oligocene	Lenticular sands	1905
		CUBA		
Bacuranao		Mesozoic	Serpentine	(1508) 1915

major synclinal folds where the sand does not carry water. A nun deep wells have been drilled without success, the deepest in 193(7586 ft., in West Virginia.

The principal interest in the region at present is the process of fl the hard, dry oil sands with water, which is generally introduced tl new wells drilled for this purpose. Flooding has been successful Bradford field, discovered in 1874, on a large, gently folded an structure which extends from Pennsylvania into New York. It of the largest oil fields in the world. One sand, the Bradford, is p tive in over 85,000 of the 105,000 acres of the field. At the close c there were 20,000 pumping wells producing 10,000 bbl. per day an additional wells which now are either abandoned or used as water ir Between 1926 and 1930 about 7500 wells were drilled. The total p tion of the field has been 280,800,000 bbl. (Dec. 31, 1929) of 4 0.804 sp. gr. oil, and it is estimated that only 28 per cent. of th the sand will be recovered by pumping and flooding. This field fourth for total recovery in the United States.

East-central States

Central and western Ohio, Indiana, Illinois, Kentucky, Ter and Michigan reached their peak of production in 1904–1908, Kentucky (1922) and Michigan (1929). In 1928, there were dri these states 2130 wells, including 680 unproductive wells, in cont 49,800 producing oil wells and, probably, 20,000 gas wells.

Central Ohio has a belt of gas fields along the western, feathe of the Clinton sand, of Silurian age, at a depth of about 2300 ft. sand is devoid of water.

Northwestern Ohio and eastern Indiana, on the northwest fl the Cincinnati arch, have a series of pools in an area 120 miles lon northwest to southeast and from 5 to 75 miles wide, called collectiv Lima-Indiana oil field. Production is from the dolomitic Trentor stone, of Ordovician age, at a depth of 900 to 1500 ft. The oil to 42° Bé., 0.843 to 0.813 sp. gr., and is chiefly of paraffin base. Ac lation is confined to porous zones in the dolomite, some of whic represent areas of subaerial or subaqueous erosion contemporaneou deposition. Concentration of the oil is on anticlines and terraces structural relief.

Another group of oil fields in the eastern interior coal basin is ce along the La Salle anticline of eastern Illinois. Oil occurs in minor and anticlines of low structural relief along the La Salle anticlir elsewhere. Production comes from limestones and sandstones, pally of Mississippian age, at depths of 350 to 2000 ft. The oil to 34° Bé., 0.880 to 0.853 sp. gr., and has a paraffin base.

Kentucky and Tennessee fields are on the north edge of the Nashville arch and on the flanks of the Cincinnati arch. More than one-half the production of Kentucky has come from the flanks of the Irvine-Paint Creek uplift, to which the Irvine-Paint Creek fault in the eastern part of the state is parallel. The oil occurs on anticlinal folds of low structural relief, principally sands and limestones of Mississippian and Devonian ages. Porosity, as well as structure, controls accumulation in limestones. The depth of production is 600 to 1600 ft., the density 38° to 48° Bé., 0.833 to 0.786 sp. gr. The base is paraffin. Gas is now being produced in large quantities in eastern Kentucky from shale of Devonian age.

Very little oil has been produced in Tennessee. The principal horizons are Trenton limestone and limestones of Mississippian age.

Michigan became a factor in oil production in 1928–1929 with the discovery of the Saginaw, Muskegon and Mt. Pleasant fields. Oil occurs on anticlines in limestones of Devonian age and sandstones of Mississippian age. The depth of the Devonian production varies from 1600 to 3600 ft., the density of the oil from the Traverse formation of Muskegon, depth 1600 ft., is 36° Bé., 0.843 sp. gr. with 0.33 per cent. sulfur content.

Kansas

Early developments in Kansas were in the "shallow" fields, less than 1000 ft. deep, and the sands were lenses and shoestrings. All were of Pennsylvanian age. Among the oil-producing formations the Bartlesville sand was the most famous. Gas is produced both above and below and, in some places, in the Bartlesville.

Discovery of Eldorado in 1913 started the important oil development of the state. Oil is found on a large anticline in Pennsylvanian sands, but the gusher production was obtained from the truncated surface of the Arbuckle limestone ("Siliceous lime") of Ordovician age (Fig. 3). This buried fold is part of the buried "Kansas granite ridge" (Nemaha Mountains) which extends from Nebraska into Oklahoma. Eldorado is the northernmost field. South of it are the Augusta, Churchill and Oxford fields in Kansas and the Braman fields, Thomas field, Garber field, Lovell field and Oklahoma City field in Oklahoma. Accumulation in most of these anticlinal fields is due wholly to structure. In a few instances the presence of buried topographic hills as well as of unconformities has permitted accumulation.

Several parallel lines of anticlinal folding trending north-south and in part bounded by normal faults have been found by the drill in Kansas, and eventually oil will be developed along many of them and on secondary lines of folding between them. Oil and gas in these folds have been found in sandstones and cherts of Ordovician and Mississippian ages and in sands and cherts of Pennsylvanian age, especially at the unconformable

surface upon which Pennsylvanian rocks were deposited in transgressive overlap toward the west. New gas fields are being developed in the southwestern corner of the state from sands of Permian age at a depth of about 2800 feet.

East of the Nemaha Mountains oil is produced from prolific fields in Greenwood and Butler counties in "shoestring" (lenticular) sands, correlated with the Bartlesville sand, which have a northeast-southwest trend and most of which probably represent offshore bars. Other productive sand lenses at right angles to the former probably represent stream

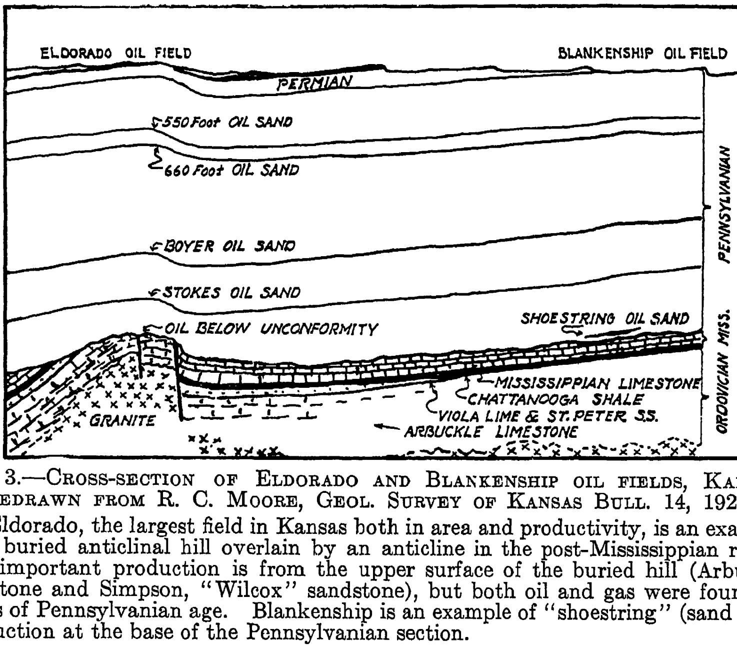

FIG. 3.—CROSS-SECTION OF ELDORADO AND BLANKENSHIP OIL FIELDS, KANSAS. (REDRAWN FROM R. C. MOORE, GEOL. SURVEY OF KANSAS BULL. 14, 1924.)

Eldorado, the largest field in Kansas both in area and productivity, is an example of a buried anticlinal hill overlain by an anticline in the post-Mississippian rocks. The important production is from the upper surface of the buried hill (Arbuckle limestone and Simpson, "Wilcox" sandstone), but both oil and gas were found in sands of Pennsylvanian age. Blankenship is an example of "shoestring" (sand lens) production at the base of the Pennsylvanian section.

channel deposits. These sands are crossbedded and carry thin coal streaks. They are sharply delimited on either side.

There were 18,900 producing oil wells in the state at the end of 1929 and the average daily production per well for 1929 was 5.8 bbl. The depth of new fields was 3000 to 4500 ft. Oil and gas have been found over most of the state.

OKLAHOMA

The Arbuckle and Wichita Mountains, extending from east to west, divide Oklahoma geologically; therefore the oil fields are designated as being in "northern" or "southern" Oklahoma. The oil fields thus far discovered are in the eastern half of the state, where the Prairie Plains homocline, consisting at the surface of strata of Pennsylvanian age,

extends westward from the Ozark uplift over which Mississippian limestones are exposed. Production has been extended southward nearly to the Arbuckle Mountains, westward to Oklahoma City and northwestward past Enid.

In southern Oklahoma most of the fields are south of a line connecting the Arbuckle and Wichita Mountains. This part of the state has been intensively prospected.

Development of oil in northern Oklahoma spread from the discoveries of oil in Kansas in shallow wells drilled for water. Few seepages of oil or gas are known in either state except in southern Oklahoma. Early wells were completed in sands of Pennsylvanian age at depths ranging from 500 to 2000 ft. The depth of drilling increased as development extended west and southwest, from 2600 ft. at Cushing to 4200 ft. in 1926 at Seminole and 6500 ft. in 1929 at Oklahoma City, and wildcat wells are projected (1930) to depths of 7500 ft. or more in search for oil in rocks of Ordovician age.

There are about 300 oil and gas fields, 3000 gas wells and 61,000 oil wells in Oklahoma (1929), and the state has produced one-fourth of the total production of the United States. It is estimated that 136,200 wells have been drilled in Oklahoma, of which 97,100 produced oil, 10,700 gas, and 28,300 were dry holes. The cost of drilling the wells is estimated to have been $4,000,000,000. About 2000 oil and gas wells are abandoned and 3500 new wells drilled annually. The daily production varies greatly. In the Nowata district, 20 years old, there are 12,645 oil wells yielding an average of 0.47 bbl. per day. In the Seminole district many new wells flowed steadily for weeks at the rate of 10,000 bbl. per day. The Seminole City field alone has 3720 productive acres, which will yield an ultimate recovery averaging 27,000 bbl. each, from 20 to 40 ft. of sand. The average daily production per well for the state was 11.3 bbl. in 1929. The average gravity of oil in the Mid-Continent producing area is 35.7° Bé., 0.845 sp. gr.; that in the Seminole area 39° Bé., 0.828 sp. gr.; at Healdton, 30° Bé., 0.876 specific gravity.

Northern Oklahoma

Production in northern Oklahoma can be classified as sand lens and anticlinal. The older fields, Nowata, Bird Creek and Glenn Pool, are examples of sand-lens accumulation. The fields are defined on the east (up the dip) by pinching out of the sand and on the west (down the dip) by salt water or change in sand conditions. The large bodies of Bartlesville sand represent deltas or current-swept banks at or below sea level. Some of the rivers that flowed westward can be traced by shoestring sands, which yield oil and gas. Burbank is now the most famous pool of the sand-lens type. Dutcher sand (of basal Pennsylvanian age) production near Bristow is also of this type.

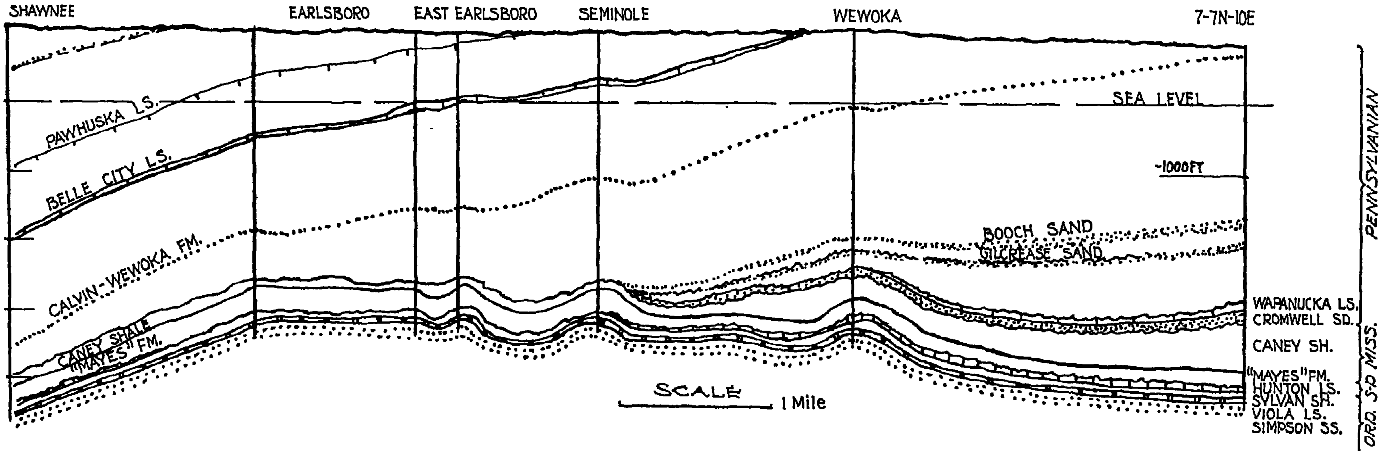

FIG. 4.—CROSS-SECTION OF THE SEMINOLE PLATEAU. (DRAWN BY JESS VERNON, OF SHAWNEE, OKLA.)

The Seminole Uplift is a structural plateau in rocks of Ordovician age north of the Arbuckle Mountains, central Oklahoma. The anticlines in Ordovician formations have accordant summits. They are reflected through the overlying unconformities into the Hunton limestone (Silurian-Devonian) and into the formations of Mississippian age, but the structural plateau is masked by convergence in the Pennsylvanian. Owing to a rapid convergence to the west, the anticlines can be identified with difficulty in the overlying shales and thin sandstones of Pennsylvanian age. They are reflected at the surface as gentle noses. The important oil production is from the Simpson formation ("Wilcox" and other sands).

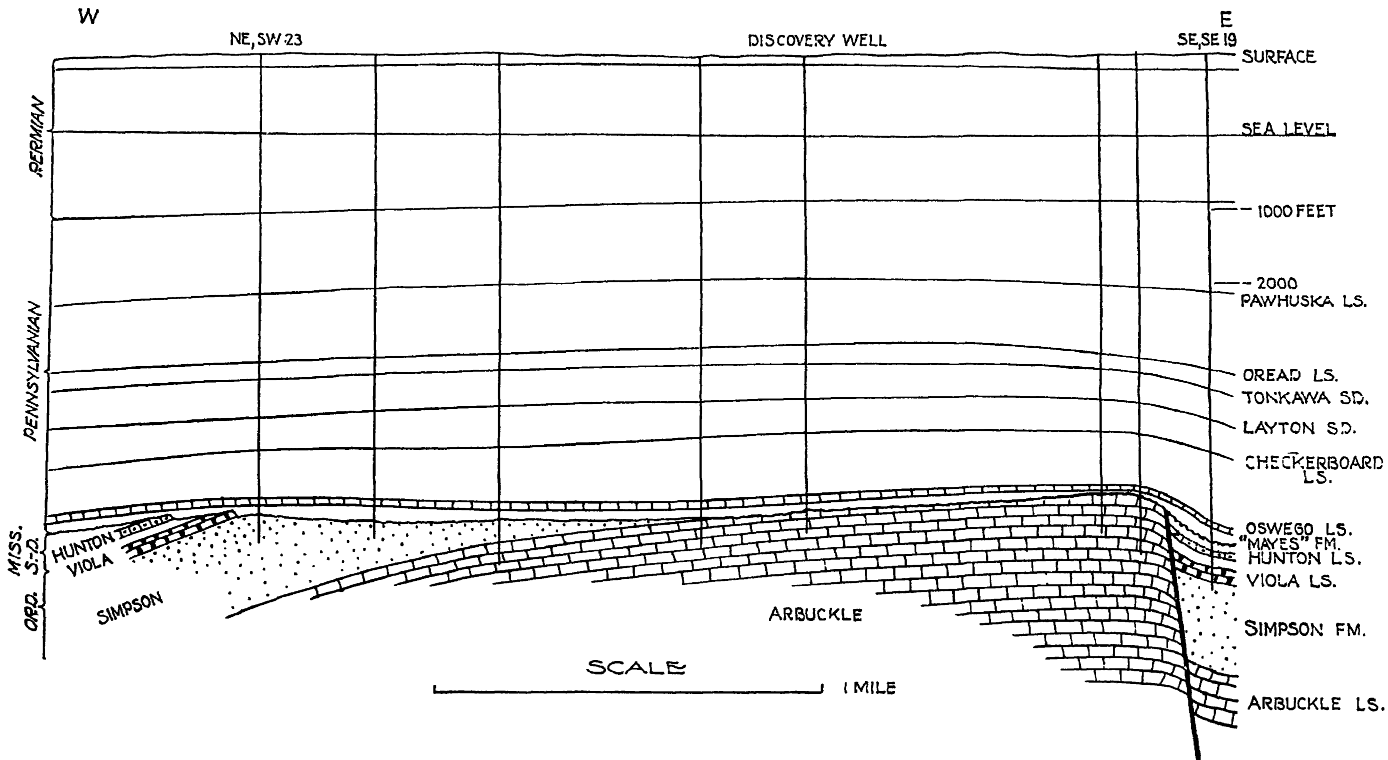

FIG. 5.—CROSS-SECTION OF THE OKLAHOMA CITY OIL FIELD, OKLAHOMA. (DRAWN BY JESS VERNON, OF SHAWNEE, OKLA.) A buried anticlinal, faulted ridge of truncated Ordovician rocks is overlain unconformably by shales of Pennsylvanian age. Oil is found in the Arbuckle limestone and in sands in the overlying Simpson formation. This field is a buried structure on the Nemaha Mountains.

More important production has been found in anticlines along old lines of folding, which were formed before Mississippian deposition and truncated and refolded several times during later Paleozoic time. Cushing, Cleveland, Ponca City, Blackwell, Tonkawa, Garber, Lovell and Oklahoma City are of this type. The Tucker sand and possibly some of the so-called Bartlesville sand of the first two fields is the Simpson formation ("Wilcox" sand) and Arbuckle limestone ("Siliceous lime") of Ordovician age. Sands of Pennsylvanian age produce in most of these fields.

Anticlinal accumulation is also found in numbers of small domes scattered irregularly throughout northeastern Oklahoma. The location of these domes may be connected with buried hills of pre-Cambrian rocks. Production from many of them is confined to sands of Ordovician age and the producing area has an average extent of only about 10 wells, but the yield per acre is as high as 35,000 bbl. Larger domes of the same kind with production from sands of Ordovician age are found in the prolific fields near Seminole on the Seminole Uplift, north of the Arbuckle Mountains, developed in 1926–1930 with gushers yielding as high as 14,000 bbl. per day and maintaining enormous production because of air lift (Fig. 4).

During 1928–1931, the Oklahoma City field was developed on the extension of the Nemaha Mountains of Kansas in sands belonging to the Simpson formation and in the Arbuckle limestone of Ordovician age below the truncated surface of a buried anticlinal ridge of pre-Pennsylvanian rocks. Truncation cut across the Arbuckle limestone on the top of the buried anticline. In 1929 one well produced a record flow of 42,500 bbl. of oil in 24 hr. from the Arbuckle limestone. This field promises to be the most productive thus far discovered in Oklahoma, to cover an area of 16 square miles, and produce 250,000,000 bbl. (Fig. 5).

Probably one-half of the total production of Oklahoma has come from rocks of Ordovician age, although the "Wilcox" sand of the Simpson formation did not become famous until 1923, when wells at Tonkawa were deepened from the six Pennsylvanian producing horizons to the "Wilcox." The Ordovician age of part of the Cushing production was not known until 1926.

Southern Oklahoma

Underground geology south of the Arbuckle and Wichita Mountains is complicated by unconformities due to mountain building early and late in Pennsylvanian time during and after the deposition of the oil sands of Pennsylvanian age. The location of some of the fields with gentle dips is determined by buried hills along old mountain rims. Other fields are on long, narrow, steeply folded anticlines.

Healdton and Hewitt, the most important fields, discovered in 1913 and 1918, respectively, overlie buried hills and production is largely

from lenticular sands of Pennsylvanian age. North and South Duncan and near-by fields represent sand-lens accumulation controlled in part by low anticlinal folding. Fox, Graham, Sholem Alechem, and Cement are on narrow anticlinal folds. Gas occurs in long anticlinal folds near Chickasha and Walters. The average depth of production is 1100 to 1600 ft. at Healdton; 2000 to 3200 ft. at Hewitt.

Chickasha and Cement are commonly thought of as being in southern Oklahoma but actually they are north of a line between the Wichita and Arbuckle Mountains. Oil has been found in the latter field to a depth of 5000 ft. in sands of Pennsylvanian age and gusher production is expected from the Ordovician rocks beneath.

Texas

Texas is divisible geologically into several oil-producing provinces—the Gulf Coast in which Recent and Pleistocene sedimentary rocks are at the surface, the Interior Coastal Plain, to which is assigned arbitrarily the part of the Plain where Cretaceous and Eocene strata are at the surface, north-central Texas where Pennsylvanian and Permian strata are exposed, and western Texas, where gypsum, red shale, salt beds and limestones of Permian age are overlain over broad areas by rocks of Triassic, Cretaceous and Tertiary ages. The "High Plains," which cover part of West Texas, are a constructional feature of Tertiary age.

Oil was found in Texas near Nacogdoches in 1838, and in wells drilled for water at Corsicana in 1896, both localities being in the Interior Coastal Plain. Until the discovery of Spindletop in 1901, Corsicana was the only producing field. Petrolia, in north-central Texas near the Red River, was discovered in 1904, gas at Amarillo in the Panhandle of Texas (northwestern Texas) in 1918 (oil in 1921), and Big Lake in western Texas in 1923. The peak of production for the state may not have been reached. The oil on the Gulf Coast is generally heavy, 22° Bé., 0.921 sp. gr.; at Spindletop, 19° to 32° Bé., 0.939 to 0.864 sp. gr. The gravity of oil at Mexia is 37° Bé., 0.838 sp. gr.; Burkburnett, 39° Bé. 0.828 sp. gr.; Yates, 30° Bé., 0.875 sp. gr. In salt domes the gravity of oil varies with depth and at Barber's Hill at a depth of 6400 ft. oil of 34° Bé., 0.853 sp. gr. is now being produced. Oil in the new fields of western Texas contains a high percentage (1.5 per cent. or more) of sulfur. At the close of 1929 there were in Texas 34,309 oil wells (9 per cent. were on the Gulf Coast) yielding an average daily production of 23.8 bbl. per well.

Gulf Coastal Plain

Discovery of oil at Spindletop in 1901 led to the development of the Gulf Coast in Texas and Louisiana over an area 350 miles long and 150 miles wide. A number of salt domes and a few domes without discovered salt were found in succeeding years because of topographic

elevations (Fig. 6), seepages, "paraffin dirt" and outcrops of cap rock. Many new salt domes have been discovered since the introduction of geophysical work late in 1922. In 1921 there were 44 known coastal domes—28 in Texas, 16 in Louisiana. About 69 coastal salt domes were known on Jan. 1, 1927—42 in Texas and 27 in Louisiana, including 9 found by seismographs and torsion balances. During the succeeding years many additional domes have been found, most of them discovered by the seismograph. At the end of 1929, it was reported that 88 domes were known in southern Texas, of which 35 were productive, and 70 were known in southern Louisiana, of which 24 were productive. One interior dome in Texas is producing and one dome in Louisiana east of the Mississippi River is producing. A number of domes have been found not underlain by salt at the depth thus far drilled, of which several produce oil. Besides oil and some gas, sulfur is recovered from five of the domes and salt from a few.

Little is known about the tectonics of these salt domes because the original salt bed from which the salt has flowed is probably buried 20,000 ft. or more. Most of the domes are circular, a few are elongate anticlines. Some are faulted, many are connected by fault lines, and some are believed to be connected at depth by salt ridges. Individual domes have had periods of growth and of subsidence.

Enormous yields of oil are obtained from these domes. The Lucas gusher at Spindletop (Fig. 7) in 1901 flowed about 75,000 bbl. per day from the cap rock at a depth of 1139 ft. In 1926 oil was found in deep lateral sands at Spindletop and the wells had initial productions as great as 12,000 bbl. per day. During the year, 13,121,000 bbl. of oil were produced from 117 wells on 34 acres, or 375,000 bbl. to the acre. One lease of 1¼ acres yielded 1,386,859 bbl. from nine wells during nine months of 1927, the average life of the wells being six months. The total production of the field to the end of 1929 was 108,117,000 bbl. from 350 acres, or 309,000 bbl. per acre. Oil sands have been found at Spindletop as deep as 5100 ft. The Abrams well at West Columbia had an initial daily production of 25,000 bbl. and yielded 1,700,000 bbl. of oil in 5½ months from 40 ft. of sand. Two acres in this field have yielded over 950,000 bbl. each. The total production of the Humble field, the largest in area on the Coast, from both cap rock and lateral sands, was 103,224,000 bbl. from 3500 acres to the end of 1929, or 29,000 bbl. per acre.

Oil is found in sands above the cap rock, in cap rock (gypsum, limestone, anhydrite) and flank (lateral) sands which pinch out against the salt. In the deep domes the sands may be continuous over the top. Flank sands have yielded most of the oil.

Production has been found from a depth of a few hundred feet (in cap rock) to over 7400 ft. The depth limit of production is controlled

FIG. 6.—AERIAL PHOTOGRAPH OF THE SOUTH LIBERTY SALT DOME, TEXAS. (EDGAR A. TOBIN AERIAL SURVEYS, SAN ANTONIO, TEXAS.)
Location of salt core at depth of 1000 ft. is shown by dashed line.

by the economic limit of deep drilling. It is confidently expected that many deeper producing horizons await discovery, because Cretaceous rocks, from which comes all of the oil in northeastern Texas, northern Louisiana and southern Arkansas, have been found in only one well (1929). Geophysical exploration is searching for domal structure at 8000 ft. (1930). Hence the Gulf Coast is a perpetual oil reserve.

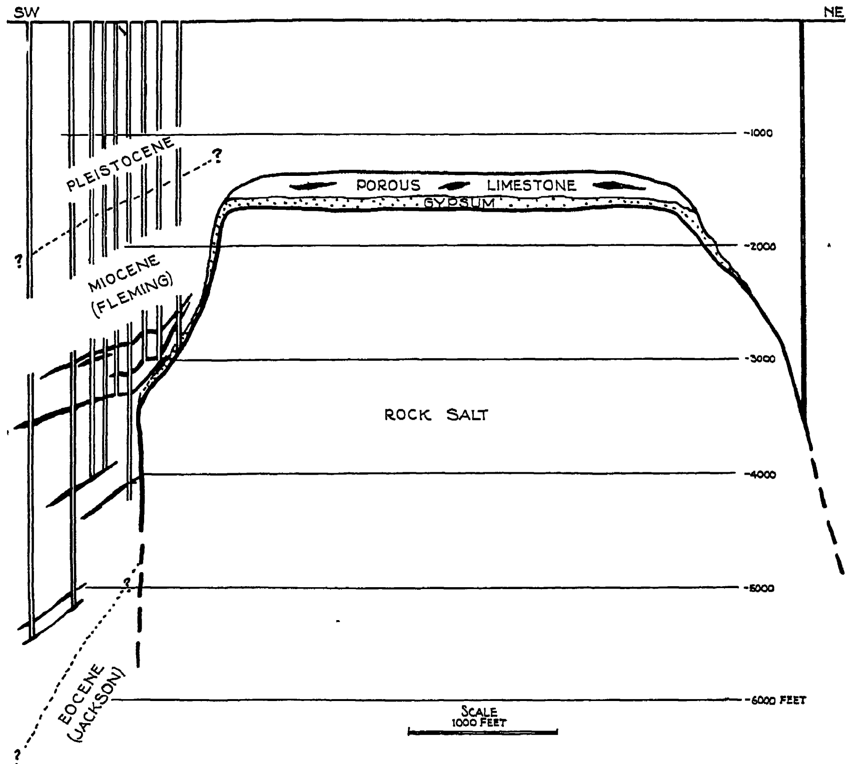

FIG. 7.—CROSS-SECTION OF THE SPINDLETOP SALT DOME, TEXAS. (DRAWN BY ALEXANDER DEUSSEN OF HOUSTON, TEXAS.)

The Lucas gusher of 1901 in the cap rock of this dome started oil development on the Gulf Coast. Oil is now produced from flank sands as well as from the porous limestone cap rock. The total productive area is only one square mile, but 108,000,000 bbl. of oil have been recovered. The section is drawn to the same horizontal and vertical scale to show the steep sides of the salt core.

The Gulf Coast province includes also many oil and gas fields between Houston and Brownsville within 100 miles of the Gulf. These fields are structurally gentle anticlines or noses underlain by thin, lenticular sands. Accumulation is influenced by sand conditions and faulting as well as folding. The age of production is upper Eocene, Oligocene and, nearer the Coast, Miocene. The most important oil fields are Raccoon Bend, Refugio and Pettus. There are many gas fields.

Interior Coastal Plain

Oil and gas production in the Interior Coastal Plain is largely from the Upper Cretaceous on the upthrow side of normal faults. The principal gas fields were in the Nacatoch sand and are now exhausted, but gas is produced in considerable volume with the oil from the deeper Woodbine sand. Discovery of oil at Mexia in 1921 beneath an old anticlinal gas field led to the development of many famous fields, among which are Mexia, Powell, Wortham, Luling, Darst Creek and Mirando. Many other fields of similar type will be discovered. The anticlinal structure at Mexia proved to be low closure against a normal fault on the upthrow side and all the other "fault-line" fields have similar structure. Oil is found in the Woodbine sand at Mexia, Powell, Wortham and near-by fields, in the porous top of the Edwards limestone (Lower Cretaceous) at Luling, and in sands of Eocene age at Mirando and associated fields near Laredo.

The Mexia fault is a zone of echelon faulting south and east of the Balcones fault, which extends from west of San Antonio northward past Sulphur River and thence northeastward probably to Stephens and Smackover, Ark. The graben between the principal faults, 5 to 15 miles wide, is cut by numerous minor faults. The throw of the Mexia zone faults is down on the west and northwest and is in some places 200 to 400 ft. The fault plane slopes to the west and northwest, so that production lies in a narrow belt between the outcrop of the fault at the surface and the projection at the surface of the trace underground where it cuts the producing horizon.

Very large production is obtained from the large fault-line fields. Mexia, discovered in November, 1920, had produced 86,730,000 bbl. from 3720 acres by the end of 1929, with 396 wells still producing, or 23,300 bbl. per acre. Powell, discovered in January, 1923, had produced 102,600,000 bbl. from 2650 acres by the end of 1929, with 524 wells still producing, or 37,740 bbl. per acre. Both fields produce from the Woodbine sand at the base of the Upper Cretaceous section, at a depth of 2800 (Powell) to 3100 ft. (Mexia).

Luling, Darst Creek, and other fields which produce from the porous Edwards limestone near the top of the Lower Cretaceous, are located near San Antonio. Luling has produced one-half as much oil as Mexia from 2100 productive acres.

Oil was discovered in 1929 on a large elongate anticline at Van, east of Powell, in the Woodbine sand. This fold is believed to be one of the largest on a general line of folding parallel to Mexia-Powell.

Oil is found in several masses of serpentine, near the Mexia and Balcones fault lines, the best known fields being Thrall, Lytton Springs and Chapman. Oil occurs in the serpentine and in crevices in the Austin

chalk (Upper Cretaceous). The serpentine probably represents tuff cones and ash beds ejected in Austin time.

One interior salt dome, Boggy Creek, produces oil, accumulation being concentrated in the Woodbine sand along a fault which is tangential to the salt core.

North-central Texas

North-central Texas includes the area near Red River, underlain by the buried Red River ridge of Ordovician limestone and pre-Cambrian granite, and also the Bend Arch between the Red River ridge and the Llano-Burnett uplift. Production is from lenticular sands in the Pennsylvanian in both areas and also from porous limestones and sands in the Bend series of basal Pennsylvanian age in the latter area.

Along Red River the famous fields are Electra, a buried structure overlain by lenticular sands, discovered in 1909, in which the peak of production was about 1915, and Burkburnett, a structural dome replete with oil sands, of which the peak of production was in 1919.

Ranger, discovered in 1917, started the boom on the Bend Arch. This field, together with Breckenridge, Desdemona and other smaller pools, yielded gusher wells of short life and small wells of moderately long life. Many gushers were abandoned soon after completion.

Sands in the higher Pennsylvanian section unconformably overlying rocks of basal Pennsylvanian age on the Bend Arch produce large amounts of oil and some gas in small pools which owe their existence to sand conditions rather than to structure.

Western Texas

Amarillo was the first oil field discovered beneath the Permian salt basin, which underlies a large part of western Texas and the Panhandle and extends northward into Kansas. An enormous gas field was discovered in 1918 in arkosic sandstone on a large anticline underlain by granite, and subsequently other gas fields have been discovered. The granite is part of the buried Amarillo Mountains which form the western extension of the Wichita Mountains of Oklahoma (Fig. 8). The area of the gas field exceeds 1500 square miles and comprises the largest of the two great gas reserves in North America. Gas is being transported 1100 miles from this field. Oil was discovered in 1921 in arkose on the flank of another buried granite hill, but the field did not prove a commercial success until the discovery of gusher production in 1925 in dolomite of Permian age, stratigraphically above and basinward from the arkose. Oil and gas are found in porous horizons in the dolomite. The oil, in both the dolomite and the arkose, overlies a horizontal salt-water table. The maximum production was reached in 1926, when there were 786 oil wells which yielded an average production of 172 bbl. per well per

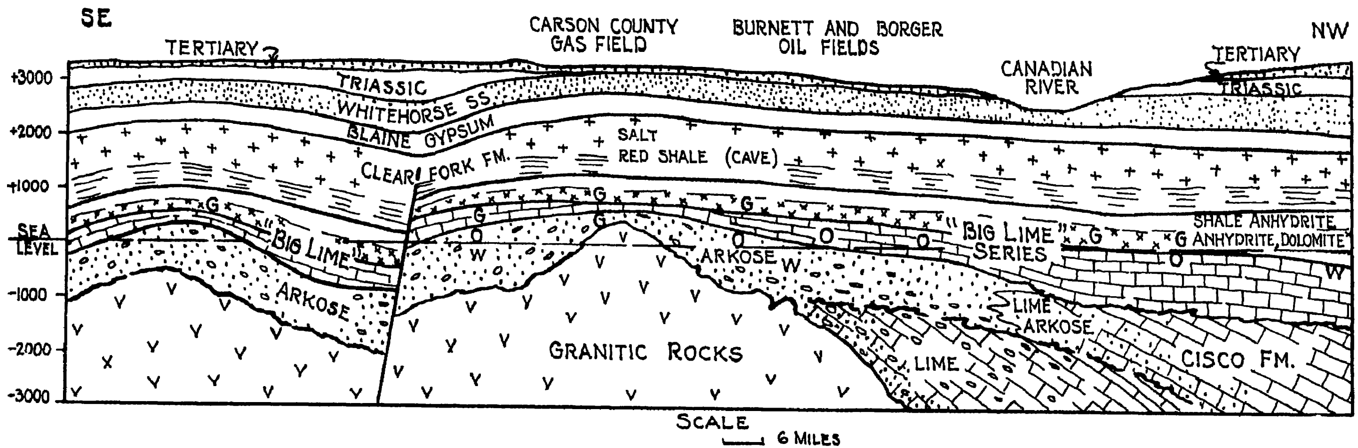

FIG. 8.—CROSS-SECTION OF THE BURIED AMARILLO MOUNTAINS, NORTHEAST OF AMARILLO, TEXAS. (DRAWN BY JOHN L. FERGUSON, OF TULSA, OKLA.)

Showing the accumulation of oil and gas in dolomite and in arkose ("granite wash") of Permian age on anticlines above buried peaks of granitic igneous rock. The Cisco limestone, of Pennsylvanian age, and an older limestone have been found by deep drilling north of the granite ridge. Helium is recovered from the gas. O indicates oil; G, gas; W, water.

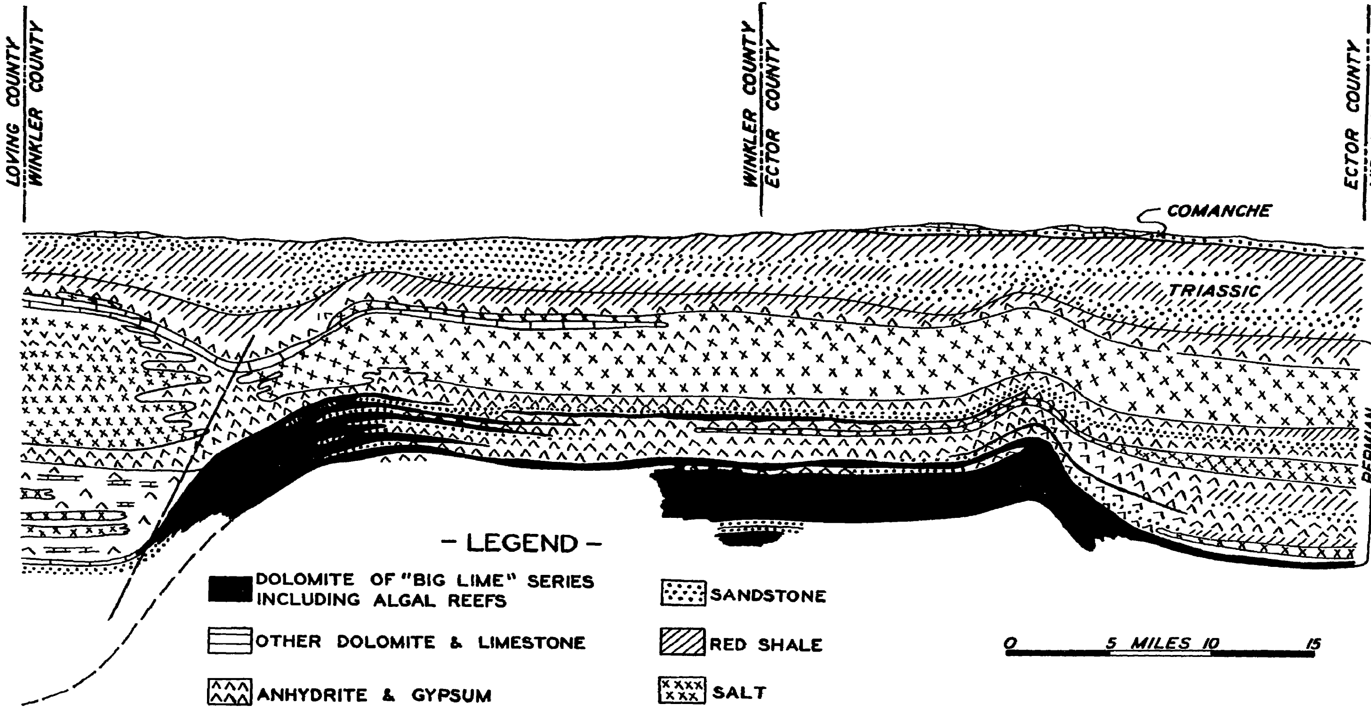

FIG. 9.—CROSS-SECTION OF HENDRICKS AND ECTOR FIELDS, WEST TEXAS. (AFTER L. D. CARTWRIGHT, JR.)

From the east line of Ector County on the east to the west line of Winkler County on the west, showing the accumulation of oil in porous limestone under salt and anhydrite beds in the Ector anticline on the east and the accumulation in porous limestone, algal reef in the Hendricks anticline on the west, especially on the west flank of this fold. The structural and depositional platform between these fields is a major tectonic feature, probably a horst, separating the West Texas salt basin on the east from the Delaware salt basin on the west. The oil is in the dolomite shown in solid black. [Reproduced by permission from *Bull.* Amer. Assn. Petr. Geol. (1930) **14.**]

day from a depth of about 3000 ft. Another peak was attained in 1929 owing to the extension of production eastward along the top and north side of the ridge in thick, porous beds of arkose between granite hills. Gas above the oil and from the gas fields is used to flow the oil. The oil has an average gravity of 37° Bé., 0.838 sp. gr., and congeals at 59° F. The gas has an abnormally low rock pressure.

In 1923 the Big Lake field was discovered near the southern end of the Permian salt basin in an oolitic limestone or dolomite of Permian age (younger than the dolomite at Amarillo) at a depth of about 3000 ft. Amarillo was discovered on a surface anticline, but Big Lake by accident, because Cretaceous limestone overlies the Permian unconformably and the reflection of underlying structure in the Cretaceous is obscure. During 1925–1927 several other fields, including McCamey, Yates, McElroy and Hendricks, were discovered near the Pecos River on the southwestern side of the salt basin (Fig. 9). Yates, producing from a depth of about 1000 ft., is one of the major and most profitable oil fields of the United States (1930). Production is obtained from porous limestone of Permian age in anticlines connected with major lines of buried structure and abrupt change in lithology due in part to the presence of algal limestone reefs. These folds extend into New Mexico subparallel to the Cordillera (Rocky Mountains). Oil is also obtained on the eastern side of the salt basin on low anticlines in sand above the limestone and in porous spots within the limestone. Many new oil fields will be found in and around the salt basin.

In December, 1927, a well was drilled at Big Lake to 8520 ft. and it has been flowing ever since (January, 1930) about 2500 bbl. per day. At that time it was the deepest producing well in the world. The producing sand is the Simpson formation of Ordovician age. The gravity of this oil is 52° Bé., 0.769 specific gravity.

Sulfur gas is associated with all the limestone production in the salt basin and sulfur water surrounds the oil in the fields near the Pecos River.

In 1929, 46 per cent. of the total production of Texas came from rocks of Permian age, although there was no production from Permian strata until 1921.

Arkansas[2]

El Dorado, in Union County, was the first field opened in the State of Arkansas (January, 1921), although the Caddo field in Louisiana, discovered in 1904, is only a few miles from the Arkansas line. Smackover, discovered in July, 1922, covers 29,500 acres and is one of the largest fields in the United States. It yielded 271,348,000 bbl. of oil to the end of 1929, or 82 per cent. of the total production of the state. Of this,

[2] W. C. Spooner, Consulting Geologist, of Shreveport, La., has kindly revised the sections on Arkansas and Louisiana.

40,580,000 bbl. was light (26° Bé., 0.897 sp. gr.) and the rest heavy (21° Bé., 0.927 sp. gr.). Production is obtained from four major and a few minor sands in the Upper Cretaceous at depths of 1900 to 2700 ft. and the peak of production for the field was 473,830 bbl. per day on May 26, 1925. Stephens, Irma, East El Dorado and Lisbon are the other fields. Heavy oil is produced at Irma and from the Nacatoch sand at Smackover and from the East El Dorado field.

Gas was found in large volume on the top of the Norphlet dome of the Smackover anticline. It is also produced from sands of lower Pennsylvanian age in the Arkansas Valley east of Fort Smith.

Louisiana

Northern Louisiana is classified in the Interior Coastal Plain region and southern Louisiana in the Gulf Coast region. Production in the southern portion of the state is from salt domes and from domal folds which are thought to be deeply buried salt domes. Fourteen salt domes are known in the northern portion of the state, but neither oil nor gas has been found associated with them. Oil was first found in commercial quantities in Caddo, northwestern Louisiana, in 1902 by drilling at a gas seepage.

The Sabine Uplift is a very large, low dome in the northwestern corner of the state, 90 miles long and nearly as broad, which existed as early in geologic time as the beginning of the Upper Cretaceous. Local folds with little structural relief are superposed on the uplift. Unconformably below the base of the Upper Cretaceous there are buried anticlinal hills of Lower Cretaceous limestone and shale. The Monroe Uplift in the northeastern part of the state is in the upper strata a similar structural feature but had a different structural history. Doming began with Upper Cretaceous sedimentation and continued nearly to the close of the Upper Cretaceous periods, and was renewed in a lesser degree in the Eocene. On its southern extension, in Richland Parish, the Eocene rests directly on the Lower Cretaceous. Several small domes with structural closure of 500 to 700 ft. lie between the Sabine and Monroe uplifts.

Northern Louisiana is underlain by sandstone and shales of Eocene age, but oil and gas were found only in the underlying Upper and Lower Cretaceous until 1926, when oil was found in the Eocene in the Urania field. Both oil and gas in the Eocene are associated with faults and low folds along a line of monoclinal folding on the south and southeast sides of the Sabine Uplift, and with a southeasterly extension of the Monroe Uplift in Richland Parish.

Oil production is obtained from the Upper Cretaceous on the Sabine Uplift (Caddo, Elm Grove, Crichton, Bull Bayou, Bellevue fields) and at Homer, Haynesville and Cotton Valley, at a depth of 290 (Bellevue) to 3000 ft., and gas is found in the Monroe field and at Sarepta, Spring Hill, Shangaloo and other fields, at about 2800 ft. Most of the "Wood-

bine sand" oil on the Sabine Uplift is now known to occur at or a short distance above the unconformity between the Upper and Lower Cretaceous. Deeper production of oil and gas is obtained from the Lower Cretaceous rocks at Pine Island (Caddo) and Cotton Valley and gas at Bethany (mostly in Texas) and in several other uplifts, at depths of 2890 from sand to about 6500 ft. from porous, oolitic limestone.

The Monroe gas field, discovered in 1916 in a sand of uppermost Cretaceous age, which produces over an area of 350 square miles, yielded 1,056,000,000,000 cu. ft. of gas to Jan. 1, 1930, from 720 wells. The estimated reserve was 2,500,000,000,000 cu. ft., total open flow 3,500,000,000 cu. ft., daily average withdrawal 470,000,000 cu. ft., of which two-thirds was used for carbon black (making 77 per cent. of the world's supply). The initial rock pressure was 1025 lb., the pressure at the end of 1926 was 784 lb. per square inch. Richland Parish, producing from a sand of Upper Cretaceous (Austin) age, has greatly increased the volume of available gas. These fields and the Amarillo fields are the largest gas reserves in North America. Gas from Monroe is piped to St. Louis, a distance of 470 miles, and to Macon, Georgia, 500 miles.

Mississippi

Gas was discovered at Jackson, Miss., in March, 1930, at a depth of about 2500 ft. in chalk of Upper Cretaceous age. The field underlies part of the city and may cover over 10 square miles. Several wells have been deepened into basalt about 200 ft. under the producing horizon, therefore it is evident that the large Jackson Uplift is underlain locally by intrusive and extrusive igneous rocks such as those exposed in Arkansas and also found by drilling in a number of wells between Little Rock, Ark., Jackson, Miss., and Franklin Parish, La. Several gas wells have had an initial production of 35,000,000 cu. ft., but the productivity of the field is yet to be determined. Gas has also been found at Amory.

Rocky Mountain Region

Oil was discovered at Lander, Wyoming, in 1833, and the first well was drilled there in 1883. In Colorado the first discovery was near Canyon City in open pits in 1864 and the Florence field was opened in 1887. Oil fields in Montana and New Mexico were not discovered until 1920 and 1921 respectively. At the present time (January, 1930) the deepest production is about 4500 feet. In 1930 oil was discovered in Pennsylvanian sandstone near Edgemont, South Dakota.

Structurally, almost all the fields are on anticlines, but not all anticlines underlain by the oil sands produce. In Wyoming 108 anticlines with closure have been drilled and 54 produce oil or gas (1928). In Montana 90 have been drilled and 11 produce. In Colorado 34 per cent., and in New Mexico 26 per cent., of anticlines with closure produce.

In all these states 268 such anticlines have been drilled and 39 oil fields and 35 gas fields have been found. The important reserves of heavy oil have not been tapped.

The largest field is Salt Creek, central Wyoming, which has been actively developed since 1908. It reached a peak of 166,245 bbl. per

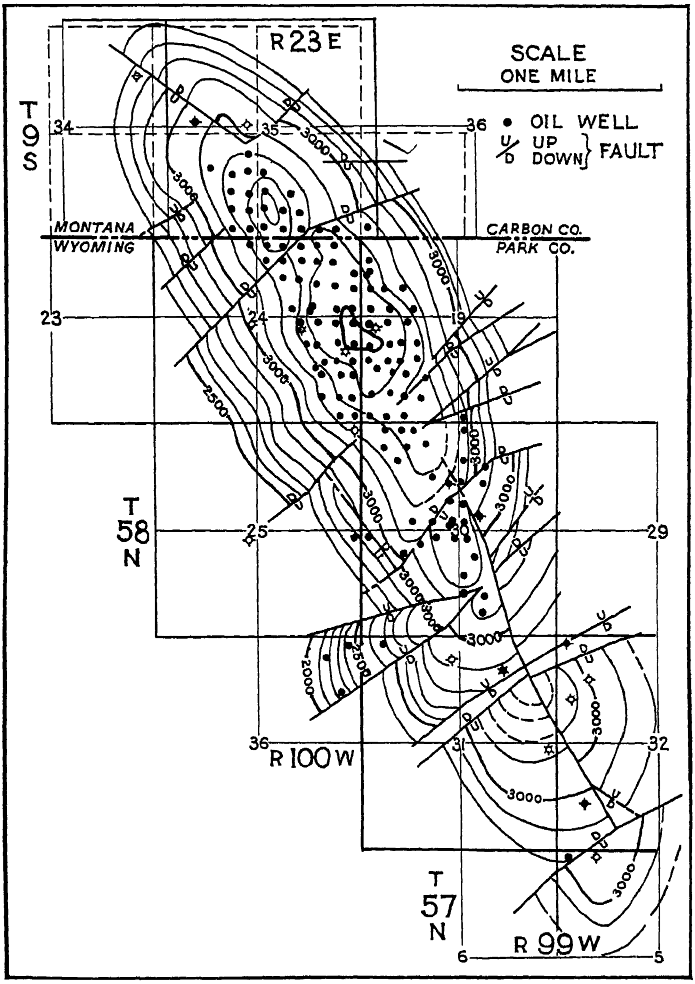

FIG. 10.—STRUCTURE CONTOUR MAP OF THE ELK BASIN OIL FIELD (DRAWN BY W. B. EMERY, OF CASPER, WYO.)

Shows radial, flank faults, some of which do not cross the axis of the anticline.

day in 1923, but the wells were not produced at full capacity for several years. There are seven producing sands in the Cretaceous and Jurassic exclusive of several producing shale horizons and of oil production in the Tensleep formation, of Pennsylvanian age, at 3800 ft. The 2127 oil wells

cover over 21,000 acres and yield two-thirds of the production in the state. Teapot Dome is a subsidiary low dome on the south end of the Salt Creek anticline, separated from it by faults.

Wyoming is the principal oil-producing state in the Rocky Mountains, with 3418 oil wells at the close of 1929 yielding an average daily production of 15 bbl. each. There are 37 oil fields and 17 gas fields, but not all of them are actually producing. The important fields are all on anticlines and domes and light oil production is largely from the Upper Cretaceous. The names of the largest fields in order of 1929 production are: Salt Creek, Big Muddy, Lost Soldier, Grass Creek, Rock River, Lance Creek, Teapot, Hamilton Dome, Elk Basin (Fig. 10), Poison Spider and Osage.

Montana has two major oil fields, Kevin-Sunburst and Cat Creek. There are several gas fields. All are anticlinal. Of the 1335 oil wells in the state in 1927, 1020 were at Kevin-Sunburst and 171 at Cat Creek. Kevin-Sunburst is a large field, on the Sweetgrass Arch, covering 18,000, acres, but sand conditions are very erratic because the oil occurs in the eroded upper surface of the Madison limestone, of Mississippian age. Another field, Pondera, was discovered in 1928 on Sweetgrass Arch.

Colorado oil fields, with a total of 193 wells (105 at Florence) in 1927, are both east and west of and within the Rocky Mountains. Wellington and Fort Collins on the east produce oil and gas from Cretaceous sands on sharp anticlines. Florence, in the eastern foothills, produces from faults and fractures in shale in a broad, low syncline. Of the domes in the northwestern part of the state, Iles and Tow Creek produce from shale and Moffat from Dakota sandstone of Upper Cretaceous age. There are also several gas fields. A carbon dioxide (CO_2) gas well spraying 45° Bé., 0.800 sp. gr. oil was completed at a depth of 5113 ft. in northern Colorado in 1927.

Oil shales of Tertiary age underlie a large area in western Colorado, southwestern Wyoming and eastern Utah. Oil-shale mining is still in the experimental stage. A small amount of free oil is obtained by mining from sands associated with oil shale.

Production in the northwestern corner of New Mexico in the San Juan Basin comes from the Dakota sandstone on sharply folded domes and the oil is very high gravity. Hogback, Rattlesnake and Table Mesa are the principal fields. Oil was found in 1929 in Pennsylvanian rocks at a depth of 6774 ft. at Rattlesnake.

Intensive search for oil in the southeastern corner of New Mexico commenced in 1928 following the discovery of the Hendricks field in Texas, about 20 miles from the state line. Geophysical work showed the northward extension of the structural belt of folding (and faulting?) overlain in places by a buried algal limestone reef. New fields are being developed (1930) along the sides of this belt at a depth of about 4200 ft. The Hobbs field, discovered as a result of geophysical work, was the

outstanding development in 1930. The Permian salt basin has been found by drilling to be divided in the deeper Permian section by the folded belt (essentially a horst?) into the West Texas basin on the east and the Delaware basin on the west. The latter is partly rimmed by algal reefs which are now exposed on the northwestern flank (Fig. 9).

Very little oil has been found in Utah. The San Juan field in the southeastern part of the state produces enough oil from a syncline to furnish fuel for further drilling. A well in the eastern part of the state near Moab produced some oil in 1926 from the Pennsylvanian on one of the newly discovered salt anticlines of this region, but production in commercial amounts has yet to be obtained. A well in the southeastern corner of the state located in the northwestern part of the San Juan Basin produced gas from strata of Pennsylvanian age.

California[3]

California has been the largest or second largest oil-producing state for many years. All the fields are south of San Francisco and many of them are near Los Angeles. Production is derived from Tertiary strata and, contrary to conditions in all other fields in the United States, the sands are so thick and closely spaced that they are referred to as "zones." Also, the rate of decline of production in many fields is extremely slow and the average production of old wells is much greater than that in the old Appalachian fields.

The most important fields may be grouped: (1) San Joaquin Valley, comprising Belridge, Coalinga, McKittrick, Midway, Sunset, Wheeler Ridge, Buena Vista Hills, Elk Hills, Lost Hills, Kettleman Hills, Kern River (Bakersfield), Mt. Poso and Fruitvale; (2) Coastal, comprising Santa Maria, Elwood, Rincon and Summerland; (3) Ventura-Newhall and Ventura Avenue; (4) Los Angeles Basin, comprising Los Angeles City, Montebello, Whittier, Brea-Olinda, Richfield, East and West Coyote Hills, Santa Fe Springs, Dominguez, Rosecrans, Inglewood, Huntington Beach, Seal Beach, Long Beach (Signal Hill), Torrance and Playa del Rey. Most of the production came from the San Joaquin Valley fields until the discovery of the new fields in the Los Angeles Basin in 1920–1923.

The San Joaquin Valley is a large synclinal basin composed of a thick section of Tertiary sediments. The oil fields occur around the southern periphery of this basin. The location of these oil deposits is determined mainly by anticlines in conjunction with unconformities and overlaps. Faulting also plays an important part in the occurrence of oil, especially on the east side of the valley. The Kettleman Hills field (Fig. 11), on one of

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[3] W. S. W. Kew, of Los Angeles, Calif., has kindly assisted in the preparation of the section on California.

three large echelon anticlines, was discovered in 1928, by a well flowing 3900 bbl. of high-grade oil or crude naphtha, from a depth of 6900 ft. It promises to be one of the largest fields of the world.

Fig. 11.—Oblique aerial photograph of northern end of Kettleman Hills anticline, California. (Reproduced by permission of Ralph D. Reed, The Texas Co., Los Angeles, Calif.) The oil field is in the center of the anticline. The Coalinga oil field, about 75 miles north, is at the base of the hills in the background.

The Coastal fields produce from sand and fractured shale, of Miocene age, folded into anticlines, modified by faulting. The Elwood, Rincon and Summerland fields are situated partly in the ocean and wells are

drilled on wharves. The anticline at Rincon was mapped under the ocean by a geologist wearing a diving suit and was later checked by airplane observation.

The Ventura-Newhall fields and Ventura Avenue (Figs. 12 and 13) have their production controlled by relatively closely folded anticlines in Miocene and Pliocene strata. Faulting is also closely associated with many of these structures.

In the Los Angeles Basin, the fields occur in a large depositional basin of Pliocene and Miocene sedimentary rocks, the fields being located around its margin and along a major fault structure which crosses the basin in a northerly direction. Most of the producing structures are domal folds, though the Whittier-Brea-Olinda production comes from a homoclinal closure against a fault. Playa del Rey is an example of production from sands abuting a buried hill.

Depth of drilling increases annually. The deepest pumping well in the world in 1926 was at Athens, Calif., 7591 ft. deep, yielding 45 bbl. per day, and the deepest well in the world, 8046 ft. deep in the Brea-Olinda field. In 1929 a well was drilled at Long Beach (Signal Hill) to 9280 ft., a new world's record. Many wells at Santa Fe Springs and a few at Signal Hill are flowing from over 8000 ft., the deepest producer being 9350 ft., at Santa Fe Springs. In 1930 a drilling well in the Midway field had reached a depth of over 9600 feet.

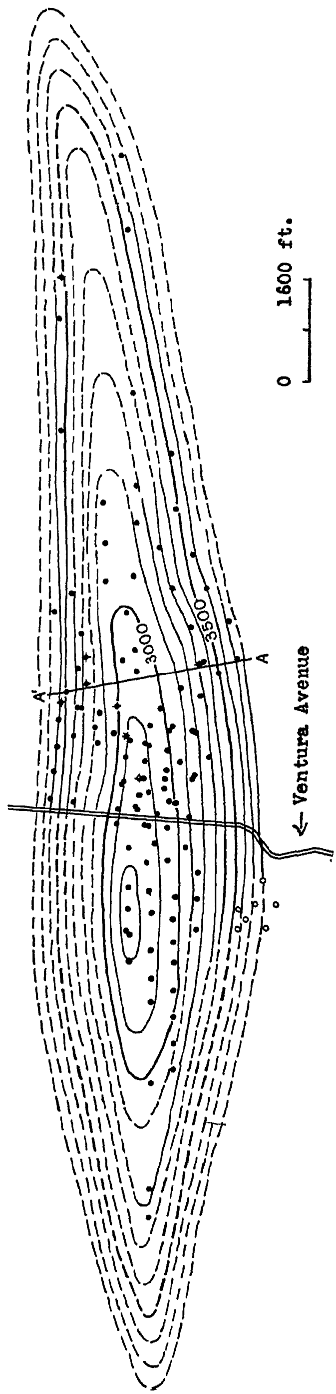

FIG. 12.—VENTURA AVENUE OIL FIELD. (DRAWN BY L. C. DECIUS, SAN FRANCISCO, CALIF.) Subsurface structure contour map. Datum below sea level. Contour interval 100 feet.

Production per acre is very high in many of the fields. For example, Coalinga, opened in 1890 and covering 15,300 acres, has yielded over 20,000 bbl. to the acre from overlapped sands without structural closure; Kern River, opened in 1899 and covering 10,300 acres, over 26,000 bbl. to the acre from overlapped and faulted sands; Santa Fe Springs, discovered in 1920, has produced to July, 1929, over 218,000,000 bbl. of oil from 1600 acres at a depth of 3500 to 8500 ft., or 136,250 bbl. to the acre from a perfect anticline. To July, 1929, Signal Hill had produced over 352,000,000 bbl. of oil from 1300 acres, or 271,000 bbl. per acre from a faulted anticline. At Santa Fe Springs 10 producing zones have already been found. At Signal Hill there are five zones, but on top of the structure the oil sand is practically continuous for a thickness of about 6000 feet.

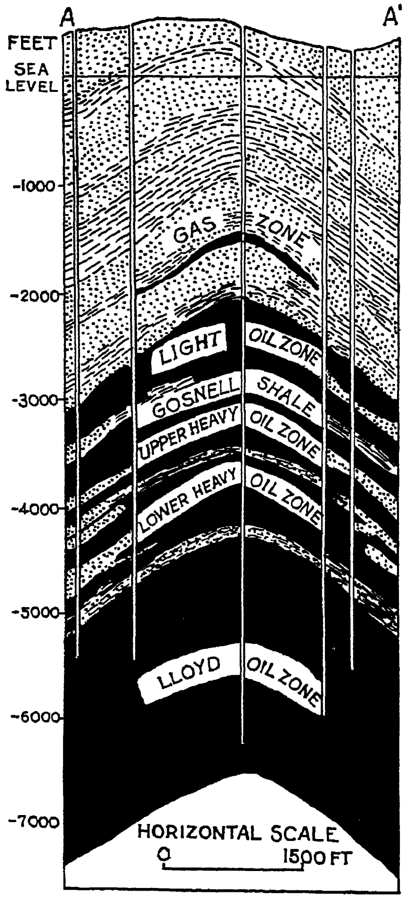

Fig. 13.—Cross-section of the Ventura Avenue oil field. (Drawn by L. C. Decius.)

This section illustrates the very thick productive oil sands ("oil zones") typical of California oil fields. Deeper drilling has discovered deeper production.

The Midway-Sunset field, discovered in 1901, had produced 669,500,000 bbl., or 13,700 bbl. per acre, to the end of 1929, from both anticlinal and overlapped sands. This field has produced more oil than any other in the United States and has produced almost half as much oil as all the fields in the rest of the world excluding the United States, Russia, Mexico, Persia and Venezuela.

Midway is also well known because of the Lakeview gusher of March, 1910, which flowed uncontrolled for 18 months and produced over 8,000,000 bbl. of oil (18° to 21° Bé., 0.945 to 0.927 sp. gr.) during this period, of which three-fourths was recovered. A second Lakeview gusher flowed uncontrolled for five months in 1914 and produced about 6,000,000 bbl. during this time.

The gravity of the oil in California varies over a wide range. It is 14° Bé., 0.972 sp. gr., in Kern River, 23.5° Bé., 0.912 sp. gr., in Midway-

Sunset, 27° Bé., 0.891 sp. gr., in Long Beach, and 60° Bé., 0.736 sp. gr., in Kettleman Hills.

CANADA

Oil has been produced in the Province of Ontario at Oil Springs since 1861 and at Petrolia and Bothwell since 1862. The discovery well of the Bothwell field was drilled on an anticline by a man who was familiar with the anticlinal theory of oil accumulation published by T. Sterry Hunt in 1861. The wells at Oil Springs are only 350 to 400 ft. deep, at Petrolia about 480 ft. deep. Production in these fields comes from porous limestone and dolomite of Devonian age. Other fields produce some oil from Silurian and a little from Ordovician limestones and dolomites. Extensive shallow gas fields produce from sandstones, limestones and dolomites of Silurian and Ordovician ages. Production has declined slowly, from 829,000 bbl. in 1894 to 120,000 bbl. in 1929. The oil has a paraffin base, is about 35° Bé., 0.848 specific gravity.

New Brunswick has yielded gas and some oil from the Stony Creek gas field, near Moncton, but the oil production in 1929 was only 120,000 bbl. Gas and oil are found in the Albert shales, of Mississippian age.

Western Canada had its first oil boom in May, 1914, when light oil was discovered on the south branch of Sheep River at a depth of 2718 ft., on the Turner Valley anticline recommended for drilling by the Geological Survey of Canada. This led to the development of one of the notable oil fields of the world. It is 35 miles southwest of Calgary, Alta., and about 20 miles east of the Rocky Mountain Front Range in the "disturbed belt" of closely folded and overthrust rocks (Fig. 14).

Turner Valley developed slowly until 1924, when Royalite No. 4 was completed at a depth of 3740 ft., about 400 ft. below the top of the Madison limestone, making 18,000,000 cu. ft. of gas and spraying 550 bbl. of crude naphtha a day. The naphtha is 73° Bé., 0.689 sp. gr. A back-pressure of 500 lb. is maintained to keep the well from freezing on account of gas expansion. In 1926, this well yielded 198,000 bbl. of crude naphtha.

Oil is produced also from the Blairmore formation ("Dakota sand," Upper Cretaceous), Kootenay formation (Lower Cretaceous) and Fernie shale (Jurassic). Most of the wells completed in these formations have an initial production of about 50 to 100 bbl. During the early life of the field many wells were completed in the Blairmore formation.

Since 1927, the development of Turner Valley has been more active because of the completion of wells in the Madison limestone from which between 100 and 1000 bbl. of naphtha per day was extracted from sulfurous gas (1 gal. or less per 1000 cu. ft.) from depths of 3510 to 5900 ft. All of this "lime" production is at least 100 ft. below the top of the limestone. In 1928 the field produced 452,000 bbl.; in 1929 it produced

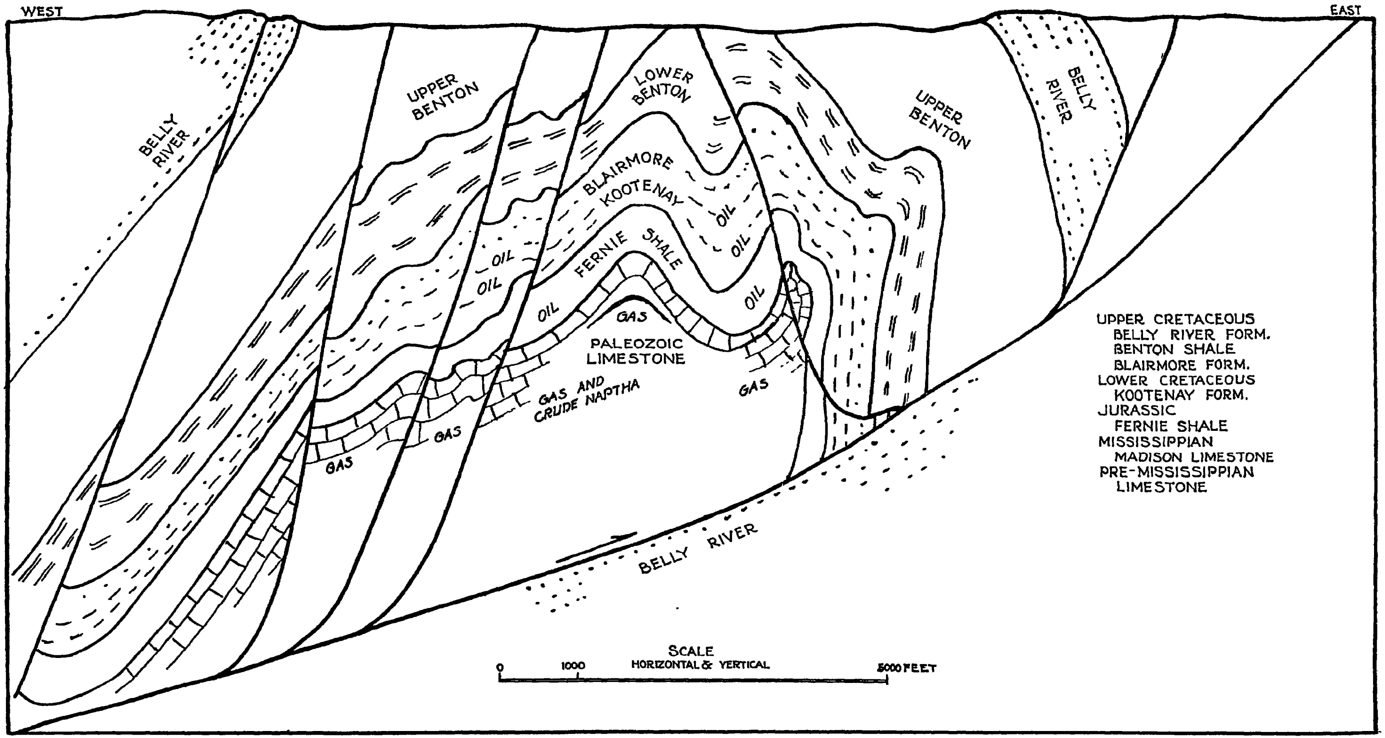

FIG. 14.—CROSS-SECTION OF TURNER VALLEY OIL FIELD, PROVINCE OF ALBERTA, CANADA. (DRAWN BY P. D. MOORE AND T. A. LINK, OF THE IMPERIAL OIL CO., CALGARY, ALTA.)

This remarkable field produces crude naphtha with large gas pressure from Paleozoic limestone as well as crude oil from overlying rocks. The surface anticline, defined by hills of Belly River sandstone, is about two miles wide, but beneath the valley there are many complex faults which form part of an overthrust ("napped") system with Belly River sandstone beneath the sole of the major thrust plane

952,000 bbl. from about 40 wells. Development always will be slow because of large lease ownerships and complicated geology.

Turner Valley (Fig. 14) is an anticlinal valley 20 miles long, carved in Benton shale (Upper Cretaceous) and flanked by strike ridges of Belly River sandstone. The dips at the surface are 40° to 70°. Subsurface geological correlations have shown that the anticline is cut by a series of longitudinal thrust faults (nappes). The Belly River formation has now been found under the Madison limestone, therefore the entire anticline is an overthrust block (nappe, Decke). The rocks are closely compressed, sheared, and possibly overturned in subsidiary anticlines with dips ranging from 45° to vertical. The structural relief on these anticlines is as great as 3000 ft., consequently the hazard of drilling even offset wells is great. The structure of this field is more complex than that of any other major field in North America. Furthermore, no oil elsewhere in the world is produced from Paleozoic rocks in such steeply folded, complicated structure.

A number of gas fields and some heavy oil fields have been developed beneath the plains of Alberta in sands of Upper Cretaceous age, but the volume of gas is small. An important gas field is being developed in the Fernie shale (Ellis formation, Jurassic) near the international boundary east of Coutts, Alta., and the Border oil field is being drilled at Coutts, north of the Kevin-Sunburst field, Montana. Production is from a depth of 2300 to 2500 ft. in the Sunburst sand (Lower Cretaceous) and Ellis sand (Jurassic).

Enormous deposits of bituminous sands at the base of the Upper Cretaceous section, where it rests on limestones of Devonian age, are known in northern Alberta, but there is no oil production.

Mexico[4]

Mexican oil fields are divisible into three groups, those on the Isthmus of Tehuantepec, the "Southern" ("Tamasopo" Ridge, or "Golden Lane") fields, and the "Northern," or Panuco fields. The Isthmus fields are connected with salt domes and production is now very small. The Southern fields extend from Dos Bocas on the north, past Tuxpam River on the south, a distance of 50 miles. From the discovery of the gusher at Dos Bocas in 1908 (which burned until the oil was exhausted and then formed a crater) until 1919, when salt water first appeared, there was a steady increase in production and the peak was reached in 1921 when new fields were developed by keen competition in drilling after many wells in old fields had gone to salt water. Panuco development reached its peak in 1923 with a production of 335,000 bbl. per day.

[4] The section on Mexico has been revised by John M. Muir, of Fort Worth, Texas, and to him the writer is also indebted for the accompanying sections.

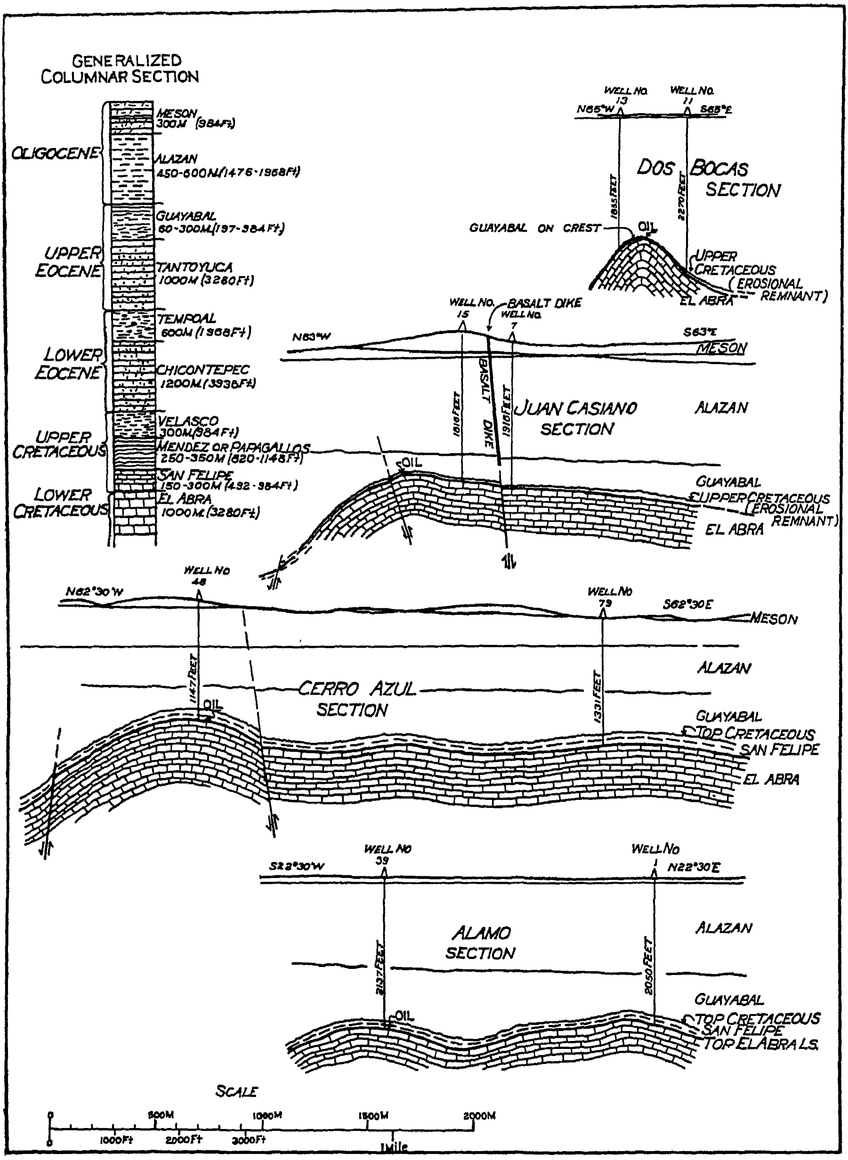

Fig. 15.—Cross-sections on the Dos Bocas-Alamo structure, the Golden Lane, or Southern fields of Mexico. (Drawn by John M. Muir, of Fort Worth, Tex.)

The Cretaceous rocks form a buried anticlinal and cross-faulted ridge overlain unconformably by shales of Tertiary age. Gusher production comes from the upper, cavernous surface of El Abra limestone. Three wells located along this buried ridge and one well at Masjid-i-Suleiman, Persia, are the four largest and most productive wells in the world.

Two of the largest wells in the world, Potrero No. 4 and Casiano No. 7, were completed in 1910 in the Southern fields. The former produced over 100,000,000 bbl. in 9 years, the latter 70,000,000 bbl., in 10 years. Both were closed in because of water incursion, which commenced in 1919–1920, but the former is again producing about 700 bbl. daily. Cerro Azul No. 4 has also made about 80,000,000 bbl. and in 1929 was still producing 2000 to 2400 bbl. daily.

The principal Southern fields are Tepetate, Juan Casiano, Southern Chinampa-Los Naranjos, Amatlan, Zacamixtle, Toteco, Cerro Azul, Potrero del Llano, Tierra Blanca and Alamo. They are on a narrow ridge of El Abra limestone (Fig. 15), a reef facies of Lower Cretaceous age (formerly called Tamasopo limestone because of incorrect correlation). This structural ridge is an asymmetric fold with a general dip of 5° to 7° on the east flank and with dips of 25° to 40° on the western flank. Through Tierra Blanca and Alamo the fold is of double nature. A series of en echelon faults cut the structure at varying angles, giving the semblance of a longitudinally faulted west flank, and many geologists have postulated a more or less continuous normal fault of great displacement on this flank. Some erosion may have taken place at the end of the Lower Cretaceous, and soon after the beginning of Upper Cretaceous time. The limestone is very fossiliferous and its high porosity is due to the fact that the fossils are represented by hollow casts or cavities. The ridge is one of the best examples of the buried hill type of structure, as the overlying Tertiary formations cover it with a more or less consistent easterly dip.

The Northern, or Panuco fields, consist of Panuco proper, Ebano, Cacalilao and Topila. A single commercial well exists in Los Esteros (or Altamira) the northernmost extension of the field north of Tamesi River. These fields collectively lie on the plunging prolongation of the Sierra Tamaulipas, which may be considered a geanticline. The most prolific production is found on the faulted and fractured flanks of the subsidiary folds of this broad, undulating arch (30 miles, 50 km., wide). The oil occurs in the lower half of the San Felipe formation and the upper part of the Tamaulipas limestone. This Tamaulipas limestone is of the same age (Lower Cretaceous) as El Abra limestone, but is a deep-water facies, nonfossiliferous, close-grained and cherty in character.

Several small fields have been produced for many years on the Isthmus on salt anticlines and salt ridges. These intrusions are not circular, as in the United States. Erratic sands have been found and the production therefore is small. The new field of Tonala, developed in 1929, has (January, 1930) several wells which had an initial daily production of 2000 barrels.

The gravity of the oil is approximately 12° Bé., 0.985 sp. gr., in the Northern fields, 21° Bé., 0.927 sp. gr., in the Southern fields and 29° to 32°

Bé., 0.880 to 0.864 sp. gr., in the Isthmus fields. All the oil has an asphaltic base.

Cuba

The Bacuranao oil field, the only one in Cuba, is 12 miles east of Havana. Production is from serpentine. About 33 wells have been drilled and seven of them produce a small quantity of oil from depths of from 300 to 1200 ft. All the wells are on 10 acres. The gravity of the oil is 27° Bé., 0.891 specific gravity.

Acknowledgments

The writer is indebted to a number of geologists for sections and other data, and is under especial obligation to those who have drawn sections and whose names are mentioned in the captions to the illustrations.

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DISCUSSION

(Sydney H. Ball presiding)

D. C. Barton, Houston, Tex. (written discussion).—From Table 4, one might infer that the gypsum-anhydrite of the salt-dome cap rock carries oil or gas. The cap-rock production is from the lime rock, which is not a true limestone but calcitic replacement of the gypsum-anhydrite. Dolomite is present in the cap only as scattered crystals.

The gravity of the crude is not quite as the figure (19° Bé.) given. The Gulf Coast "A" crude, which used to be the chief crude produced and which now composes

30 to 40 per cent of the total production, commonly has a gravity of 20.5° to 23.5° Bé. The crude which forms the rest of the present-day production has most commonly a Beaumé gravity of 24° to 32° . A few crudes have higher gravities, which range up to 42° Bé. A little crude has a gravity below 20° Bé. At Spindletop, the cap rock crude had a gravity of 22° Bé. The crude from the flank sands has a gravity ranging from 23.5° to 30.5° Bé. The Miocene and Oligocene crudes characteristically have a lower Beaumé gravity than the Eocene crude, but there is also a vertical variation of Beaumé gravity which averages 3.3° increase for each 1000 ft. of depth.

S. Powers (written discussion).—Since this paper was written the East Texas oil field in Rusk, Gregg and Upshur counties, Texas, has been developed. This field is one of the largest in area and productivity in the world. At the present writing (July 16, 1931) it is 31 miles long from north to south, 3 to 9 miles wide, and will cover about 150 square miles. The estimated ultimate yield is 750,000,000 bbl. of 40° Bé. gravity (0.823 sp. gr.) oil. Production is from a sand or a sand zone of Eagleford and Woodbine ages at the base of the Upper Cretaceous and the oil has accumulated along the eastern shore line where the sand feathers out against the Sabine Uplift. The shore line from north to south through the length of the field is approximately the same elevation below sea level. The edge of production on the north, south and west is controlled by water and the contact between water and oil is a horizontal plane which is about 160 ft. below the edge of the sand on the east side of the field. Other examples of similar shore-line accumulations are the fields along the east side of Lake Maracaibo, Venezuela; several fields along the west side of the San Joaquin Valley, California; the shallow oil fields of northeastern Oklahoma and southeastern Kansas; many of the shallow fields of western Pennsylvania and western West Virginia and the Clinton gas field of eastern Ohio.

Future Gold Production—The Geological Outlook

By L. C. Graton,* Cambridge, Mass.

(New York Meeting, February, 1931)

The Economic Perspective

Although marked by numerous well-known attributes of its own, gold does not possess a kind and range of physical, chemical and geological characteristics wholly different and apart from those of other metals. Instead, it is merely one of the family of metals, with its individual and distinctive traits subordinate to the underlying family relationships and resemblances. The geologist concerned with pure science will pursue, therefore, the same general course and will examine the same broad kinds of facts and interrelations when studying the *occurrence* of gold as when studying that of copper or cadmium, or any other metal.

But when we come to consider the *use* and *economics* of the metals, we discover that gold differs strikingly from all the others. Gold, in short, is unique, not by virtue of its inherent characteristics, but because of a peculiar and concerted psychological regard in which it is held. It will be evident, therefore, that the geologist who would deal with gold supply must give far greater weight to economic considerations than is required in his contacts with any other metal. If the geologist be called the scientist of occurrence, and the economist the scientist of use, the economic geologist stands as the necessary link between them. Perhaps it is in part because the economic geologist has made himself more familiar with the science of occurrence than with the science of use and has thus in some degree failed to bridge effectively the gap between the two fields, especially with regard to gold, that the users[1] of gold now appear to be in a state of general concern and even of alarm for the security and adequacy of future supplies.

In the case of no other mineral commodity do the users, as a rule, trouble themselves greatly about the adequacy of supply except in so far as that may affect the price which they shall have to pay for the commodity. Oversupply, with resulting low price, they regard as a gift of good fortune. As to long-term undersupply, they are generally content to go ahead on faith, leaving the producer to discover as best he may the means for meeting their requirements or the inventor to find a substitute. But history has shown that for gold a different situation exists. While

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[1] Instead of the customary terms *consumption* and *consumer*, I am here employing *use* and *user*, since these seem more suitable for a substance so imperishable as gold.

oversupply in the main is accepted complacently, the prospect of undersupply is looked upon with grave concern. The state of mind among financiers and economists now finding such general expression is not novel in quality and possibly not more insistent in tone than that which prevailed about 40 years ago, after several decades of virtually stationary gold production. It is perhaps only because the business of production, the business of use and the business of intercommunication of ideas are all more highly developed now than they were two-score years ago that the air seems fuller at the present time than ever before of the forebodings of dire consequences to our economic structure if a steadily increasing gold production shall not be maintained.

Although the user of gold and his agent, the economist, give evidence of real concern at the prospect of undersupply, and although some steps have been taken in recent years which fortunately have had the effect of using the available gold more efficiently in certain directions, is it unfair to suggest that these professional students of the use of gold are still far from applying a general remedy for the ills they describe and predict? They discuss and dispute among themselves, but to the outsider, at least, agreement appears rather in reiteration of warning than on analysis of basic causes and on constructive measures of improvement. Moreover, there seems to lie in some of their words an implication of reproach that the producers should have permitted this unhappy prospect to arise, suggesting something akin to our instinctive feeling of resentment against the city government for dirty streets or crowded schools. It is easy to draw from their presentations the conclusion that the gold-mining industry, which long was depended upon to serve the world's needs, has failed more recently to live up to its responsibilities, so that now the financiers and economists must step in to take real control.

Of course it is to be remembered that the obligation of maintaining output is no greater than that of wise use, and that when output falls despite the efforts of the producer, the need increases for intelligent economies. Real cooperation—or at least coordination—between producer and user obviously is necessary if economic and fiscal health are to be maintained. The producer, instead of realizing any dereliction on his part, is inclined to feel that for too long was the use of gold left to hit-or-miss influences and only recently and tardily are the experts of use coming to a full recognition of their part in the composite responsibility.

Some of the places where it would seem to the producer that improvements might be achieved in the concepts relating to the use of gold are suggested briefly at the end of this article. But it must be admitted, of course, that there are economic factors in production just as there are in use, and on the producer must fall the responsibility for proper understanding and control of these factors. Since efficient and economical production is intimately related to the quantity that can be produced, it is

permissible to give some attention to this question in considering the prospects of future gold supply.

ECONOMIC STATUS OF GOLD PRODUCTION

Unfortunately, the circle of interest of the average metal producer is of small diameter. If his costs of production are reasonably less than his selling price and if he can foresee, either positively or through the spectacles of his characteristic optimism, reasonable continuance of his operations, he is usually satisfied and his thoughts and efforts extend little or no further. Personal profit is his guiding motive. His part as one of the cogs in the machinery of world economy rarely occurs to him, and still more rarely influences him. This state of mind applies not only to the individual mining company, but with almost equal effect to gold-producing countries: if their gold-mining industry is thriving, if they can collect from it and from those dependent on it a satisfactory slice of taxes, they are content and give little concern as to whether there is or is not enough gold to go around. If the gold-producing countries adopt (as a few of them have done in some minor degree) any policy looking toward stimulation of gold production within their boundaries, such stimulation is not primarily for the purpose of increasing the available gold supply for the trade of the world, but rather for the immediate and selfish purpose of increasing their own proportion of this available supply.

Such narrow and self-centered attitude on the part of the producing units and the producing countries applies especially to gold, above all other metals, through the very circumstance that makes gold psychologically unique. With other metals, the restricted and selfish attitude of the producer finds its own approximate corrective in the fluctuations of metal price, and every substantial producer of such metals is thoroughly and constantly aware of this tendency to bring production into equilibrium with demand. Inadequate production brings increase in price and thus a tendency working through self-interest to enlarge the production.

No equal stimulation to individual or national productive endeavor takes place in relation to gold. With its guaranteed market and its "constant" price, gold too easily causes its producers to forget that its price is fixed only in a mathematical sense while in the economic sense, *i. e.*, in purchasing power, which is by far the more significant, the "price" is extremely variable. Stated in another way, the relative advantage in self-interest to the producer of gold in times of low general prices is a far less obvious and direct influence than is the direct stimulation of other metals by increase in their respective prices. It is evident that there is far greater elation in the copper industry, the wheat industry or the leather industry caused by a 20 per cent advance in the prices of those commodities than there is in the gold industry by a 20 per cent fall in the general level of commodities and wages. This is in spite of the fact that

virtually the whole of the 20 per cent change is likely to be velvet to the gold miner whereas only some fraction of the 20 per cent changes represents increased profit to the producers of other commodities.

There is, perhaps, some natural basis for the tendency of the commodity producer to place greater importance on selling price than on cost of production, even though he may fully recognize in his saner moments that a change in the one is just as vital to him as in the other. The reason for the difference in his instinctive appraisal of price and cost may lie in the fact that cost in considerable measure is within his control whereas ordinarily price is something that he cannot directly influence at all. External tendencies toward increased costs, therefore, he feels he can partly or wholly offset and consequently they do not so greatly depress him as external tendencies of equal magnitude toward decreased selling price, and on the contrary, decrease in cost resulting from outside influences seems to him less occasion for jubilation and optimism than equal increase in selling price.

It may be held that all this is merely a question of state of mind, and therefore not directly concerned with the practical question of gold supply. On the contrary, the difference in attitude as regards gold and other commodities bears directly upon our subject. The steps in the reasoning may be presented briefly thus:

The major fluctuations of metal prices move in swings or sweeps of some duration, corresponding in the main with the general cyclic variations of trade and industry as a whole. There are thus commonly considerable periods of high metal prices and ensuing considerable periods when the prices are low. When the price falls, the producer's profit is reduced; if it falls far, his entire profit may be threatened. Plainly, he must do something to meet this situation. The most obvious and effective thing he can do is to bring costs down. In part, this may be accomplished by temporary expedients, mere borrowing against the future. But in substantial degree, reduction of costs is likely to be accomplished through relatively inexpensive but constructive improvements in method dug out of the human brain by the stern compulsion of adversity. Such reductions in cost for the most part become permanent and apply as well in future up-swings as in future sags of the price curve; thus they increase the economically available reserves of the metal concerned.

In periods of high metal prices, on the other hand, profits are plentiful, courage is high, the public temper is everywhere optimistic, and aggressive activity is in the air. Plainly, this is the time for the metal producer to think of expansion. Generous allotments are made for better and bigger equipment, and development work is increased to permit enlarged output. Perhaps most pertinent of all, outside exploration and acquisition, both near and far, are inaugurated with fervor and lavishness. All this likewise inevitably results in expansion of the known metal reserves and thus provides or insures the supplies for mounting needs.

Thus each major rise and each major fall of metal price brings a characteristic reaction, *both of which operate to increase the known and economically available reserves of the metal.* Moreover, *each major rise brings a direct and important increase in actual output,* especially as compared with the repressed or perhaps deliberately curtailed rate in times of low metal price.

But gold experiences no such influence. In times of general prosperity, the gold miner finds sledding the hardest; while the margin of profit is thus small he sees little appeal in the idea of buying or hunting for more gold mines or of expanding his own production (which had not previously undergone curtailment). On the other hand, when the business cycle is in the depths and the gold miner is the only one really in clover, he is likely to find a public wholly apathetic to any kind of exploratory project and he, himself, is likely to fall partial victim to the contagious pessimism then prevailing around him.

In consequence, I believe that in these modern days there is, decade by decade, less intensive search for gold occurrences than is true for at least several of the metals that experience marked fluctuations of price. With the passing of exploratory search more and more from the hands of individual prospectors into the control of strong organizations, the peculiar advantage possessed by gold through past ages in stimulating the cupidity of searchers has now largely if not wholly been lost, since a dollar's worth of profit in iron or nickel looks as good to a corporation as a dollar's worth of profit in gold. And with the passing of this advantage, which operated to stimulate gold hunting in the past, has come this more modern lethargy toward gold hunting, which arises from the somewhat artificial assumption of a more level economic keel for the gold industry than for the semiprecious metals.

Whether enough people equipped to search for ores will come to recognize that there is just as great justification to speed up the search for gold in times of general depression as to speed up the search for other metals in times of general prosperity is something better to be settled by the psychologist than by the economic geologist. But unless that state of affairs comes about, or some equivalent tendency, it may be expected that the proportion of gold finding to available gold to be found will lag behind the corresponding proportion for those metals for which search is especially stimulated in times of great economic activity.[2]

[2] The foregoing conclusion probably holds true even though periods of high prices for the metals of variable price (and periods of low prices for gold) may *not* be really the best time to search for new deposits. With the tendency toward steady enlargement of mining units, the growing interval between discovery and the beginning of production may often bring new production on the market just when it can least well be absorbed. But this maladjustment is not likely to be eliminated easily; doubtless exploration will continue to be conducted chiefly while the public is in a speculative mood, even if the eventual benefits should be less than by a better timing of effort.

It is perhaps partly because of the conditions just discussed that some uncertainty appears to exist as to the effect of general prices on gold production. "All the available evidence," says a recent studious report, "seems to indicate that gold production is remarkably insensitive to moderate changes in the price level."[3] What apparently has been taken as support for this view comes in part from the mining industry itself. For instance, in the report just cited, the head of one of the largest lead-silver producers answers in the following words the question as to how much of the recent decline in gold production of the United States is due to increased operating costs and how much to exhaustion of deposits:

"We have heard a great deal of the shutting-down of gold mines because of the rise in the cost of labor, operating costs, etc. In most cases, however, the best ore in these mines has been exhausted and what is left are the low marginal ores. Any material change (increase)[4] in costs will shut down such mines . . . In the main, the decline in production is due to exhaustion of profitable ores."

The structure of the foregoing answer would seem to imply that the decline has come rather from exhaustion than from rising costs; but its content indicates exactly the opposite. The existence of "marginal" ores discloses immediately their dependence on the spread between cost and selling price, a dependence further emphasized by the fact that additional increase of costs would necessitate closing down; and the exhaustion of the "profitable" ores of a deposit is of very different significance from the complete exhaustion of a deposit as a geological entity.

And in the same report, the directing head of a great copper-producing group, in replying to the same question, indicates that decreased operating cost is not likely to have much effect on "primary production from gold ores, strictly speaking, because vast tonnage deposits of low-grade gold ores are not common, as is the case, for instance, with copper ores." But in this reply, the "marginal" ores in the existing mines appear to be mainly ignored. Yet nearly every gold mine has these marginal ores, available only if and when costs are low, and differing from marginal ores of other metals only in that the latter may also become available, without decline in costs, through increase in metal price.

On the other side of the picture, the Director of the U. S. Geological Survey, who is in annual statistical contact with all the gold mines of the country, is quoted in the same report as saying: "In the United States, higher operating costs were responsible for a large part of the rapid decline in production since 1915," . . . "but exhaustion of the more easily mined and higher grade orebodies has also been a considerable factor."

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[3] First Interim Report of the Gold Delegation of the Financial Committee, League of Nations (September, 1930) 13.

[4] Parenthetical insertion by L. C. G.

Still more definite is the statement by the Government Mining Engineer of South Africa, likewise in this same report; this official, intimately familiar with conditions on the Rand, says of that great district: "A fall in the cost of production of 2 shillings a ton will maintain the output at its present level for about 10 or 12 years." That is to say, he concludes that a decrease of about 11 per cent in costs would prevent what, on the basis of present costs, has been forecast by others as a decline in Rand production of over 40 per cent for the tenth year hence, and of 53 per cent for the twelfth year, or of 4¼ and 5½ million ounces, respectively.

It thus seems impossible to doubt that the general level of prices and wages, through influence on production costs of gold mines, affects the output of gold in just the same way that the output of other metals is affected. But it is doubtless true that the effect is more restrained for gold than for the other metals, and perhaps more influential in the less conspicuous way of preventing or postponing declines and exhaustion than in the more direct and obvious way of causing positive increases in production.

A real census of the probable quantity and tenor of "marginal" gold ores would be interesting and possibly of real significance. At present, knowledge on the subject is very sketchy.

One final consideration may be pertinent under this heading. In the United States, in particular, but to some extent in many parts of the world, there has been heavy mortality among the small mines and corresponding ascendency of the large units. This is often referred to as if it were a fashion or style which better fits the megalo-psychology of the time than would the more modest basis of earlier years. But the relative disappearance of the small mine rests on something more fundamental than habit or fashion. High labor costs of the present century have forced mines more and more into extensive mechanization. But mechanization is not justifiable or efficient except for operations of substantial magnitude. For most of the metals, large deposits exist and have been developed or discovered in sufficient number to permit mechanization almost universally and still maintain all the needed production despite the low tenor of most of these great deposits. This has had the direct effect of making competition by the smaller mines extremely difficult if not impossible, and it has had the same effect indirectly by drawing the best talent to the larger units. But the geological conditions of gold deposition appear to have been such as to produce a relatively smaller number of great deposits suitable for the effective utilization of wholesale mechanization. Those that are now known are not supplying all the gold that the world appears to need. At the same time, the small gold mine is about as inefficient and as destined to economic death as the small mine of other metals, and thus is unable in any material degree to reduce the indicated shortage, unless it fortunately possesses ores distinctly richer than average.

Magnitude and Trend

It is unnecessary to repeat here lengthy or detailed assemblages of statistics relating to gold production. But it may be useful, in order that the essential background of the past may be brought close at hand as a basis from which to extend a few forecasts, to recall certain basic statistical facts.

If we follow the conventional procedure of beginning the consideration of gold production with the discovery of America, we find that in a general way the world production of gold may be divided into four periods: A 360-year period of less than one million ounces annually, a 40-year period of about six million ounces annually, a 15-year period of increase from six to twenty million, and finally a 25-year period averaging in the neighborhood of twenty million ounces per year.

If one were to look merely at this generalized curve of production, he would naturally be inclined to predict that the next substantial change would be a new surge upward. But there are, of course, several obvious factors which fail to support such a prediction. In the last few years, one or another of these adverse factors has been dwelt upon by various students, with the resulting conclusion, widely concurred in, that not only will the future curve of production fail to show any such upward jump as those of about 1850 and 1890 respectively, but that the curve will even fail, and fail badly, to hold its own at present levels for more than a very few years to come. Many, indeed, predict a strong decline (held to be already in evidence) beginning by 1940, or before.

It would certainly be the easiest thing and perhaps the safest thing to follow these almost unanimous predictions of impending major decline in gold production. While impressed by the quality of most of the arguments adduced to support such predictions, and while realizing keenly the danger of plodding against the general tide of opinion, I feel that the *quantitative* aspects of certain of these factors have been somewhat overdrawn. This does not mean that I dare forecast any substantial increase from present levels of production, or that decline from present levels may not be inevitable. It does mean, however, that so far as my own foundations for guessing are concerned, I would prefer to wager that any material decline is likely to begin farther in the future and to cause a tapering off more gradually than the current predictions of others seem to imply.

Analysis of Factors

The factors which are emphasized by present prophets of declining production include:

1. The bare statistical fact that the output for the past few years has been notably lower than the maximum output prevailing a few years previously.

If we disregard the abrupt sag for the years 1917 to 1923, inclusive, occasioned by withdrawal of man power and the general economic turbulence of the war and by a severe labor strike on the Rand, we find that, for the years 1925 to 1930, gold production has been running at about 90 per cent of the average for eight years ending with 1916. Even if it be assumed that all depressing effect of the war on gold production is now behind us, a 10 per cent fluctuation either way in an output coming from so many sources and produced under such a wide range of conditions cannot be taken, from the statistical standpoint alone, as indicating a permanent change to a new and lower level of production. Fluctuations of this relative magnitude have been common enough in the past and they must be admitted as natural possibilities for the future. Of course, a 10 per cent drop makes the decline *in ounces* conspicuous indeed when the going annual level is in the neighborhood of 20,000,000 oz. But after all, it is *relative* change that is important, not the absolute change in ounces.

The decrease in United States production to less than one-half of what it was 15 years ago and the decrease in Australian production to less than one-seventh of what it was 25 years ago are regarded by present-day prophets as highly significant. Of course, the greater the relative importance of the countries that show decline, the more probable is a falling future output for the world as a whole. And inasmuch as the United States and Australia have been great producers, the effect of their decline is undoubtedly serious. But we must not forget that dominance in gold production has waxed and waned for one country after another during all the time for which records exist. Egypt, Hungary, Spain, Colombia, Brazil, Russia, the United States, Australia, the United States again, and finally the Transvaal, each has for its own time held first place in gold production, and each in turn (except the one now fortunately in that position), having reached its zenith, has been displaced. But during all this time, gold production has, on the whole, increased. To assume, therefore, that decline in relative rank of important producing countries in recent years is an indication of some new order of things seems quite unsupported by past history.

Whether the high general price level prevailing during most of the past 15 years is a direct consequence of the enormous gain in gold output during the preceding 25 years, or is due to destruction of property during the war, or represents a widespread inoculation by the psychological germ of inflation, or perhaps is some combination of all these influences, the fact remains that no regression *in general prices and wages combined* has yet reached such low levels as would tend in important degree to stimulate gold output through notably lowered costs of production. Therefore, from the purely statistical standpoint alone, it appears unsafe to assume that the present somewhat depressed output may be expected to continue falling henceforth, *unless we expect prices and wages to stay where*

they now are or go higher. As a matter of fact, it is *decline* of present price and wage levels that is said to be threatened by lowered gold output. If there is valid ground for this view, there must be admitted a tendency to stimulate gold production if such decline in price and wage levels should become at all acute.

2. It is held by many that because of the ages-old search for gold by man, and especially because of the gradual subjugation of the remote places of the globe, the chance of finding new discoveries to take the place of the steady exhaustion of known occurrences has become so slim that maintenance of output at approximate levels of the present cannot long be expected.

That the world is becoming steadily better known is obvious and that each discovery lessens the total number remaining to be discovered is equally evident. *Sometime* it will become true that the rate of new discoveries fails to make good exhaustion of the known occurrences. The arrival of that time, however, will not be demonstrated at the time it arrives, but only well afterward, when the permanent trend of statistics will have become unmistakable. To assume that such time is at hand is nothing novel for the present epoch. Similar doleful assumption was made and apparently widely believed about 1890. But we know now how far from the truth it was.

Unless, therefore, it can be shown that at this particular time as distinguished from all previous time we have actually and at last reached a break in the curve of new discoveries, we are little better justified in dreary predictions now than were the financiers and economists of 1890. Three considerations, however, do better justify a present prediction of future decline than any corresponding prediction made earlier: (1) The mere passage of time; the longer production persists, the nearer, of course, we are to eventual exhaustion; (2) the magnitude of production in recent decades has been enormous; during the last 25 years as much gold has been produced and absorbed as in the preceding 400 years—this shows what terrific inroads we are making in the total eventual supply, and how impossible it will be to keep up this rate indefinitely; (3) to my mind most important is the fact that the backbone of present production comes from a single district, the Rand, a district whose output is so much greater than that of any other district previously known to the world that the prospect of discovery of another district of corresponding magnitude seems improbable. One index of the preeminence of the Rand is the fact that no country except South Africa itself has in any year yielded as much as one-half the output of the single Rand district. Great countries like the United States and Australia, each with many important producing districts, fall far short of the product taken from that one occurrence a few tens of miles long; and since 1923 that district, singlehanded, has each year produced more than all the rest of the world put together. Therefore,

when production from the Rand really begins to decline, world production will also decline unless an unusually large number of new districts of more conventional magnitude shall be discovered whose collective yield will offset the decline on the Rand. This possibility, too, seems improbable.

Thus we reach the conclusion, amply evident anyway, that the fate of gold production, at least for several decades to come, is largely dependent upon what the Rand will do. Right here is one of the places where, in my judgment, a more pessimistic opinion now prevails than is justified by the facts and probabilities. To this I shall wish to revert in somewhat more detail on a later page.

3. The third factor emphasized by prophets of declining production is the conscious or unconscious assumption that the economic efficiency of production will remain in the future about where it is at present. This is equivalent to holding that human ingenuity, with respect to gold production, has about attained a dead level, from which no material improvement can safely be counted on for the future. To my mind, this is a serious mistake, counter to all past experience. This subject may be considered under the three heads of metallurgy, mining and exploration.

In metallurgy, it is true that the efficiency of gold recovery has reached a distinctly satisfactory state, with the cyanide process for "dry" ores, with efficient concentrating and smelting for the copper ores, the development of selective flotation for complex ores and (in less general degree) the utilization of dredges for placers. The proportion of gold now being lost from the ores treated must be regarded on the whole as satisfactorily low. Likewise, on the whole, metallurgical costs are altogether creditable. But it is to be remembered that the consequences of further improvement in either recovery or cost are not limited to the ores now being treated. Each gain in recovery of 10¢ per ton not only increases the gold output directly to that extent, but also makes available for profitable treatment tonnages which were previously unprofitable, but which may yield when treated many times the 10-cent improvement. Decreased costs are similarly effective. Recognizing this, metallurgists are not content in resting on the fine achievements already attained, but are continually seeking ways of squeezing out from the tailings or slags a cent or two by this means, a cent or two by that, and in pushing costs ever lower. No one is justified in predicting that the opportunities still remaining are so slight as not to be economically worth while, or devoid of effect on world output.

In the mining operations, I feel confident that very important improvements are possible for the great bulk of the gold output. It is, of course, as erroneous to assume that what the Alaska Juneau Company has done can be duplicated by the rank and file of other gold producers as it is to assume that what the Miami Company has done is a fair index of mining costs per ton of copper ore the world over. But here, at least, are con-

spicuous and enviable attainments which other producers will be impelled to emulate. Conditions that affect mining vary between such wide limits from district to district, especially with regard to the physical and geometrical characteristics of the ore itself, that no such standardization of mining efficiency can be expected as has been achieved in metallurgy. Because of this wide variation in local conditions and the consequent absence of a fairly uniform standard of what mining costs should be, there is lacking in mining much of that competitive stimulation to achievement which is so evident in metallurgical costs. And while the metallurgical efficiency may be stated and compared in precise terms of percentage recovery, only vague and indefinite means exist for recording the efficiency of mining operations.

In a general way, the problems of mining are wrestled with by the local staff alone, while to the problems of metallurgy all the world contributes assistance. Metallurgy is eminently adapted to laboratory research, and profits accordingly; but we have not yet learned much (if we ever shall) about solution of mining problems on any other than the full scale of the mine itself. On this great scale, experimentation is so costly as to be rigidly limited; and experience and wisdom accumulate slowly. It is inevitable, therefore, that, by and large, mining is less well done than is metallurgical extraction; one needs only to move from mine to mine in different districts or on different continents to realize how true this is.

For other metals, the forward steps in improvement are likely to come, as already implied, under the stern compulsion of periods of low metal prices, and once attained are pretty effectively held henceforth. But the nominally constant price for gold brings no such direct compulsion to improve costs. Bad work can continue to be regarded complacently if a fair profit is realized; how much greater the profit might be is relatively ignored. Therefore, lower grade gold ores lie longer untouched, and the tendency toward increased output is less.

In the course of time, with the trend toward greater standardization of underground methods and costs and the increasing custom of journeys of observation by mine officials, marked improvement in mining costs for gold may be expected, especially through the further introduction of mechanization.[5] In my own opinion, no important district offers greater promise in this regard than the greatest district of all, the Rand; and in no possible way can the world's gold output be so profoundly benefited as by a substantial reduction in Rand costs.

Turning now to the subject of exploration, we find it convenient to divide this topic into two sections—exploration in going mines for the

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[5] A striking and peculiarly successful example of coping with natural and economic handicaps is afforded by the adoption of air refrigeration, first on surface, and more recently underground also, by the Morro Velho mine in Brazil.

extension of known orebodies and the finding of new ones, and exploration for the discovery of wholly new mines and districts. In both of these divisions, improvement is both possible and actual. Better work and more favorable results are being secured in the detailed geological work conducted in producing districts than in the more rough and ready generalized efforts of exploration in the broader sense. The great increase in understanding of ore occurrence resulting from intensive geological research in an old mine like the Homestake is notably improving efficiency and success in the search for more ore in that leading producer of the United States. Results such as these, experienced in a number of great mines, are equivalent to the outright discovery of brand new mines of substantial magnitude. The success of highly detailed geological investigations of this nature fully justifies their employment in restricted areas already demonstrated by abundant past production to possess in high degree the conditions favorable to ore deposition.

Unfortunately it is impossible to maintain that equally rigorous and scientific geological study on an appropriate scale is justified at present in the search for new mines in virgin regions. It is to be remembered that most mining districts have come into being through the discovery of ore deposits (or their weathered products) actually outcropping at the surface. This is almost as true of discoveries made in recent years as it was decades or centuries ago. And, obviously, no great store of geological knowledge is required for such discovery. In short, it must be admitted that scientific geology has as yet contributed little to the world's metal reserves in so far as the finding of new districts is concerned. Only when geology shall become able, through scientific analysis and diagnosis of the surroundings, to locate orebodies of which no direct expression is visible at the surface can a new era of exploration in the broader sense be expected. That achievement of this kind is possible is abundantly proved in the case of petroleum, of which enormous, even embarrassing, supplies have been disclosed by geological study of regions where not the slightest indication of oil seeps or other direct evidences were to be found at the surface.

We may grant that the conditions governing petroleum accumulation are far simpler than those which determine the deposition of metallic ores, and thus that prediction of subsurface oil pools will always be simpler than corresponding location of "blind" or nonoutcropping orebodies. Nevertheless, there seems no room for doubt that continued inquiry into the causes and conditions of ore deposition and into the compositional and structural rock environments favorable therefor will steadily increase and improve the power to find ores that are not exposed. The ability in this direction already demonstrated through intensive application of modern geological science in established districts gives some measure of the promise that may lie ahead in regions as yet virgin.

It cannot be doubted that great quantities of ores must exist within the range of depth now attainable by mining operations but not yet exposed to view by the present stage of erosion. If the ore geologist can attain only a fraction of the proficiency and success in subsurface prediction that his brothers in petroleum geology have already achieved, the beneficial effect on metal resources may be great indeed. At present, two handicaps stand in the way of such an outcome; first, as already implied, an understanding of the physicochemical and structural influences governing ore deposition that is as yet only partial and imperfect, and second, the lack of economic justification for applying the present geological skill with sufficient concentration and intensity to extensive areas. The first of these difficulties is in process of removal; from the immediately practical point of view, the rate of progress may seem disappointingly slow, yet when one looks back over the last 30 years, he finds that enormous strides have been made in establishing a sound and useful philosophy of ore deposition and ore occurrence. Progress in this direction is accelerating rather than standing still or retrograding. The second handicap will diminish in proportion as reliable elimination can be made of truly unpromising great areas, so as to bring within the range of economic practicality the more intensive type of geological effort upon selected smaller tracts. This, too, demands better geological science, but it also finds promise from the directions now to be mentioned.

In any present-day consideration of mineral exploration, two modern methods require comment; namely, aerial prospecting and geophysical prospecting. In connection with the search for metalliferous deposits, the airplane has thus far chiefly been used either to facilitate transportation or to discover actual outcrops of ore. In the former of these uses it has, of course, been highly effective for remote or difficult country, though few actual ore discoveries, whether of gold or of any other metal, yet stand as the direct result of improved transportation by air. The more immediate use of the airplane for locating actual outcrops or other features directly connected with ore occurrence has not yet lived up to the promise it was believed by some to hold out; it may well be, indeed, that its service in this direct way will never be great. But between the one extreme of mere covering of the ground—transportation—and the other extreme of trying to locate individual deposits, there lies an intermediate realm of potential usefulness of the airplane which may prove to be highly important in certain types of country. By inspection from the air, and particularly by means of aerial photography of considerable areas, it is already possible to gain ideas of large-scaled geological relationships and structures which would be revealed if at all only by relatively careful and expensive mapping of great areas on the ground. Improved expertness in the interpretation of geology from the air may well lead in the future to the effective and sufficiently economical selection of loci where the geological

conditions are such as to warrant the concentrated application of detailed search on the ground for individual ore deposits.

As for geophysical prospecting, others have pointed out that gold ores as such are ill suited to discovery by geophysical methods. While agreeing with this conclusion as it applies to methods now known, I do not feel that it is safe to assume that the approximate limits of geophysical applications are now in sight. Even though I have no overpowering enthusiasm for the kind of geophysical prospecting that is now possible, whether for gold or for any other product, nevertheless I have too much faith in human ingenuity to exclude the possibility that next week or next year someone may devise methods that may be highly successful, even perhaps with gold ores of certain types. To hold open such possibilities seems no more visionary than to have entertained the possibility of a cyanide process or the art of flotation.

Geological Types of Occurrence

Viewed in a rather broad and practical way, four types of gold occurrence are to be recognized as important: (1) the placer deposits; (2) rich veins or lodes associated generally with effusive igneous rocks (lavas) and falling in the shallow-seated type of deposits, according to the classification by Lindgren;[6] (3) veins and replacement deposits associated with dominant base metals, particularly copper, and falling for the most part in the class formed at intermediate depths; (4) deposits generally of lode-like form, genetically associated with deep-seated intrusive rocks and themselves falling in Lindgren's deep-seated class of deposits.

If the conglomerate ores of the Rand be regarded as of hydrothermal origin, as I believe they should, rather than of alluvial derivation, all the important placer deposits of the world are geologically young and lie at or close to the present surface. The conditions that would cause them to be easily found and relatively quickly exhausted are evident. While I have not found definite statistics on the point, it seems probable that the greatest part of the world's gold output up to the time of the important lode discoveries in California and Australia had been derived from placer deposits. But since that time, the relative importance of these superficial sources has greatly declined so that now they contribute only about 10 per cent of the annual total.

Since the placer deposits represent mechanical reworking of previous concentrations, it is natural that deposits in the hard rocks should have been sought where placer occurrences were found. In some regions, notably in California and certain of the Australian districts, this search was abundantly rewarded. But in several other great placer regions, no relatively important yield from lodes has yet been derived. Most of the placer fields in this category lie in regions of low topographic relief where,

[6] Mineral Deposits, Ed. 3. New York, 1928.

in consequence, exposures of the hard rocks may be either interrupted by cover or obscured by deep weathering. The gentle land surface in such regions, moreover, is generally a consequence of extreme old age which has permitted levelling through long-maintained wear. Therefore very great thicknesses of rock probably have been treated in nature's great erosion mill, and thus relatively plentiful accumulations of placer concentrates could be produced from a terrain relatively lean in lode gold. It would be obviously unjustifiable, however, to go so far as to predict that search for lode deposits will be relatively unsuccessful in regions where important placer occurrence is present on old erosion surfaces of low relief.

The shallow-seated lode deposits are mainly associated with effusive rocks of Tertiary age. These effusions have been localized at places of intense crustal deformation and generally occur in mountain regions which subsequent erosion has not yet worn down. Such regions usually are conspicuous and rock exposures are plentiful. Enough is now known of the characteristics of the outcrops of these bonanza deposits to make it probable that new deposits of this kind will not be easily missed, at least in regions where any search that can be dignified by the name "prospecting" has been at all thorough. These deposits, often phenomenally rich in their upper levels, have been found to play out economically at depths only one-half (or less) of that to which mining on other lode types has already been carried. Tremendously productive while they last, they give out rather quickly and, for the most part, when once dead appear pretty surely dead for all future time. Because of the very fine size of mineral particles characteristic of these shallow-seated deposits and because of the generally unfavorable physiographic conditions of their position, these shallow-seated deposits have yielded only relatively insignificant quotas of placer gold.

The association of gold with base metals appears more important on the North American continent than elsewhere. Except in connection with copper ores, the contribution from this source is relatively unimportant. It is worthy of note—and this is a tally on the pessimistic side—that among the great copper-ore reserves brought into being in the last 20 years or so, particularly in Chile, the Congo and Rhodesia, the proportion of precious metals is extremely low. Since these reserves are destined to yield an important and probably increasing fraction of the world's total copper output for some decades to come, the relative importance of gold production from copper ores seems likely to decline. Among newer copper discoveries of prime importance, this tendency is offset only in the great Frood deposit of Sudbury, Ont. I realize that this view of declining output from copper ores is not the prevailing one. Perhaps the difference arises chiefly from diverse ideas as to whether copper production in the United States will or will not continue to expand materially.

Both the chemistry and the texture of the auriferous base-metal ores seem to have been unfavorable for the derivation therefrom of important placer accumulations.

The remaining geological class of gold deposits—lodes of deep-seated origin formed by the ascent of mineralizing solutions along permeable channelways—constitutes the great backbone of gold production at the present time. It will be evident that I am here putting into a single class deposits which in Lindgren's more strictly geological classification appear in two groups; namely, deposits of the deeper intermediate zone and of the deep zone proper. But from the practical standpoint of physical persistence of the deposits downward and reasonable constancy of grade with depth, it seems to me that we can properly group together deposits of pre-Cambrian and of distinctly later ages, perhaps into Cretaceous time. Most such deposits have undergone deep erosion, therefore the portions existing from the present surface downward were formed at depths so great that a slow rate of change in quantity and character of minerals deposited tends to be the rule. Such deposits, therefore, tend to be both relatively uniform in tenor and physically persistent in depth. All the deep gold mines of the world fall in this general class and most of the world's deep mines are gold mines. While it is not to be expected, and is indeed known not to be true, that all deposits in this geological category will enable profitable mining to be carried to great depth, the presumption is far stronger for deposits in this group than for any other that richness and size of orebodies encountered in the upper levels will persist reasonably unchanged to depths that are great by our present standards of mining; namely, 6000, 7000 and 8000 ft. vertically.

These are the deposits also in which relatively coarse particles of gold are likely to be plentiful and from which, therefore, deep erosion has best chance of yielding important placer accumulations. From the standpoint of eventual gold supply, however, it may be regarded as unfortunate that, owing to the particular nature of the permeable channelways utilized by the ore-depositing solutions, the particles of gold deposited in the Rand conglomerates were too tiny to be caught and concentrated as placer deposits when the vast overlying parts of that enormous occurrence were eroded to the present land level.

The frequency with which deposits in this group are attended by placer accumulations, likely to be the first discovered, has obviously aided in the disclosure of many of the lode deposits of this kind. But many of these deep-seated deposits are enclosed in very ancient rocks, which in the course of ages have suffered profound erosion to the stage of peneplains. On these gentle surfaces, in cooler climes, there are likely to be obscuring accumulations of tundra or other swampy vegetation, while in regions nearer the equator deep weathering may be a handicap to mineral discovery. It seems rather safe to predict, however, that this group of

deposits, now the most important, is likewise the group in which, during the next 50 or 100 years, the most important new discoveries will be made. It is also the group in which, as previously indicated, there is the best chance of commercial ore occurrence at depth. Therefore, for both present and future, these deep-seated gold deposits appear to be the dominant group.

Regional Outlook[7]

It would require far more study than I have given to this subject, and personal familiarity with a far greater number of regions than I possess, to allow important additions or reliable alterations to the existing summaries of probabilities of future gold production from various parts of the world. Therefore I shall make no attempt whatever to enter into this topic in any detail, but shall present only a few generalizations which seem justified by the facts available.

One can hardly doubt the truth of the common assertion that the older and more densely settled countries are likely to be first exhausted of their gold supplies. This is indicated as an abstract proposition and appears to be confirmed by history. Of the two major countries in which declining gold production is now looked on with alarm, the United States perhaps can be classed in the group of old and well-known countries, but this can hardly be held for Australia. Even in many parts of our own West, the traveler cannot fail to be impressed by the great, unbroken stretches of wild country where any but the most conspicuous outcrops may have escaped attention. That the best has already been discovered is doubtless true, but that much remains for discovery cannot be safely denied, especially when one recalls that the area sufficient to include a great mine occupies only a veritable pin point on the map of a township or a county. Still greater, it would seem, are the relative opportunities in Australia—a vast continent yet little opened by routes of communication and climatically forbidding throughout much of its extent, yet possessing great areas indicated to be potentially suitable for ore occurrence.

In Canada and Russia, the geological setting appears to offer much, while the conditions of exposure of orebodies are such as to favor the view that more remains to be discovered than has been found. Though I feel somewhat dubious on the point, there is a common tendency to believe that a similar conclusion holds true for Alaska. In Canada and Alaska, only the forces of nature are to be contended with, but in Siberia there is added to these the handicaps of political instability and, in places, the presence of a decidedly unfriendly population, in consequence of which the resources, whatever they may be, are likely to be developed only slowly.

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[7] I am under grateful obligation to the Bureau of International Research for contributing toward recent studies of several important gold regions which I had not previously visited.

Elsewhere in Asia, only that vague and forbidding region stretching from Outer Mongolia to Tibet holds any substantial promise of future gold output, and as to the importance of that, the available record is rather legendary than definite.

In South America, no substantial additions can be counted upon from the West Coast and the high Andes. Great areas of the lower eastern slopes of the Cordillera are still virtually unknown in detail, but apparently more favorable for gold occurrence, especially in the form of placer deposits, than are the rocks of the higher mountains. And farther east and north, the great upland plains of Brazil, the Guianas and Venezuela must still be regarded as by no means exhausted of possibilities.

Central Africa holds untested possibilities, but on present indications these apparently cannot be rated highly. The great copper province of southern Congo and Northern Rhodesia, so far as now known, appears notably deficient in precious metals. The still older rocks of Southern Rhodesia contain gold mines of such importance as to suggest that more intensive search may lead to further discoveries, though the legend that what is now Southern Rhodesia contains the renowned mines of ancient Ophir seems to have little foundation in fact. The possibilities of further discoveries in this tableland of ancient rocks comprising Southern Rhodesia and the Transvaal may perhaps be indicated by the fact that the Rand deposits, though outcropping as rusty bands rich in places to the grass roots, were not discovered until nearly 20 years after the great diamond rushes to that general region. And the mammoth diamond pipe of the Premier mine remained unknown for 35 years from the time of the first diamond discovery, although stream gravels and surface exposures in that whole area were being intensively, even frantically, examined for diamonds by hordes of prospectors.

As for the Rand itself, the future prospects seem less gloomy than they are commonly painted. In the first place, those familiar only with the tense and incessant driving for bigger and bigger output, which is so characteristic of American methods, will find difficult of understanding the moderation and hesitation with which extension of operations was carried from the older locations of the Central Rand to the no less important continuations of the same horizons farther to the east. And even after this eastern Rand has so munificently confirmed the wisdom of those extensions, there still remains a surprising apathy toward wholesale exploration still farther out from the known oreshoots. The recent indicated demonstration that the East Geduld and the Daggafontein properties are really profitable mines still seems to occasion little aggressively optimistic regard for stretches of ground still farther away. Whether the placer or the hydrothermal theory of origin be accepted, there is an inevitable degree of appeal in the question as to what may be present on the south rim of the synclinal basin (partly covered by younger rocks)

opposite to the position of the Central Rand on the north rim. Yet this has thus far stirred the South African imagination to make what seem only the most casual and apparently incomplete investigations, when the magnitude of the possibilities is taken into account. It is true, of course, that the development of a great mine on the Rand from the beginning as a prospect is a long and expensive undertaking and that numerous such efforts have proved failures. It is also to be recognized that from the standpoint of steadiness of gold supply and availability of new gold in the future, it may be fortunate that the full possibilities of the Rand basin have not been more speedily tested out. In any case, it seems clear that the estimates which have been made as to future probable tonnage, although the figures mount to very high dimensions, have been tempered in their making by the same kind of restrained conservatism as has characterized the actual operations of extending the development of the reefs.

Moreover, when one considers the great additional reserves that can be brought into existence through lowering of costs to a level that I believe to be wholly feasible, and the additional great possible reserves lying in portions of the basin not now credited with any gold content, it seems necessary to conclude that the Rand may long outlast the present current estimates of its probable life and that its eventual marked decline in production may be postponed materially beyond the dates now predicted.

The fact that here the deepest mining in the world is done has both optimistic and pessimistic implications. It implies on the one hand that a number of the mines that have been mainstays of the past have been exhausted through the greater part of the vertical range that will be physically accessible to man. But it implies, on the other hand, assurance of the ability and of an established technique of following these auriferous reefs down their converging limbs to perhaps as great a depth below the surface as man will ever attain in any part of the world. And there is likely to be no other region ever found that will yield so much gold per thousand feet vertically.

Finally, when we come to the Islands of the Sea we may note the recent disclosure of rich ores reported to be of most promising extent in the Bulolo field in the southeastern tip of New Guinea. Although this island has long been known to contain gold, discoveries that are heralded as of first importance were not made until shortly after control under mandate of this portion of the island by Great Britain, which already controls over 70 per cent of the world's output of gold. If there is significance in this, let it be hoped that Britain may do as well in others of her mandated territories.

Lay Reflections on the Use of Gold

In the last 80 years, the world's annual production of gold has been increased nearly twentyfold, to a present level of about 20,000,000 oz. per year. This has not happened by accident or good fortune. It has not

come as the reward of merely reasonably intelligent and reasonably persistent effort. Instead, this great achievement has been won only through struggles, hardships, dangers and sufferings wholly unknown amid the tapestries and mahogany of legislative halls, cabinet chambers, counting rooms and college libraries. Mining is a stern business which demands red-blooded, courageous, resourceful and clear-thinking men. Men of this stamp have wrung from the earth her mineral riches, and in so doing have in addition brought about for the common good of civilization new lines of world communication, new avenues of trade, new opportunities for worthy human endeavor, new contacts, new ideas, new inspirations. To do this they have paid a heavy price—Arctic cold and tropical fever, solitude, combat, starvation, these have been the legal tender of payment, with work of the most arduous kind constituting the common currency of the transaction.

It is hardly to be wondered at, therefore, that the gold-producing industry as such is inclined to feel that, up to the present, at least, it has fully discharged its part of the total responsibility, and is disposed to leave to others a part of the worrying about the adequacy of gold supply. Neither is it surprising that the gold producers look with something of mingled indifference and annoyance on the failure of the prophets of gold shortage to achieve greater agreement on constructive programs of amelioration or offset. While realizing that any such program, to be effective, must have widespread acceptance and support, gold producers may be forgiven if they wonder whether so many of the proposals thus far made have not as yet failed of general backing, either through lack of genuine merit or because interests, hobbies and prejudices of cliques or of nations have been put ahead of the common good.

The gold miner may be pardoned, furthermore, if he feels that, in time of need, the metal which he works hard to produce is used or at least distributed in ways that, under the circumstances, seem to him unnecessary and extravagant. Some of the questions likely to hover in his mind and on which he would be glad to have enlightenment may perhaps be stated in such terms as the following.

When the interchange of goods and services between nations has such vital influence on the health of international trade, and when the productivity and thrift of its people and the quality of loans and investments of its banks so determine the internal financial strength of a nation, is it really logical to hold that possession of some arbitrary quantity of gold is a life-and-death matter?

If the prime objective of monetary policy is the maintenance of a national currency at a standard level—that is, parity with gold—has it been unquestionably demonstrated that possession of an arbitrary reserve of gold is indispensable to this end?

Has proportionate immunity to the effects of the present world-wide depression been parallel to the relative magnitude of gold reserves held by

the several countries and independent of other conditions and influences of which the gold reserve may be only an expression?

Have prices risen during the last two years in those two countries which have greatly increased their already great gold reserves while most other countries have been exporting gold?

Can the deflation of prices since 1925 be fairly ascribed to gold scarcity, when an average of close to 10,000,000 oz. has each year been added to the monetary stocks?

If it be granted that there is not enough gold to meet the supposed gold requirements of all nations, is it desirable and preferable that such gold as does exist should be held in proportion to these national requirements? If so, is there any reason to believe that such proportional distribution, if once made, would be at all permanent? If immediate redistribution to something like present proportions would be likely to ensue, is it reasonable to suppose that the situation would be materially improved if only there were a larger world total of available gold?

Can a nation, powerful because of the marketability of its great inherent resources, see long-run wisdom in such an extreme tariff policy as forces its foreign customers to pay largely in gold, and therefore ever more largely in gold—until purchases must inevitably cease? And can other nations believe that indulgence in a similar policy will give to them also power and "protection" in any real sense?

When great nations, in Asia and elsewhere, for ages have conducted their trade largely on a silver basis, is it level-headed wisdom to encourage, press and all but force these nations to some kind of gold standard at the very time when the low gold supply is so loudly lamented and when, in consequence of this and other acts of the gold enthusiasts, silver has been put so under suspicion as to fall to a price far lower than ever has been known before, with resulting hardship to a great industry and virtual demoralization of Oriental trade?

Has the nation which, with its dominions, produces over 70 per cent of the world's gold but an insignificant portion of the world's silver been influenced on this account to assume the leadership in the displacement of silver by gold, only to discover that possession of gold through production does not insure permanent retention of it nor the supposed advantages that attend great gold reserves?

Even if the abstract merits of a universal gold standard be granted, is it common sense and prudence to persist blindly toward that end in the face of a threatened decline in gold supply, and in spite of the cornering (whether wisely or not for them) of about 60 per cent of the available supply by two countries, which apparently can corner still more?

Who knows and how can it be demonstrated whether a ratio of gold to currency and credits of 40 or 30 per cent or any other figure is the safe and wise one? Would reduction of the now legalized reserve ratios in the

various countries really achieve the intended purpose of making the available gold supply "go farther," or would such action have the effect of wholesale stimulation of extravagant credit extension in directions that would defeat the object sought?

Are commodity prices positively proved to be so directly dependent on gold production and so independent of all other variable factors that the indicated rough relationship between price level and gold output, for the several decades following general adoption of the gold standard by the industrial nations until the war, can be taken as certain measure of what must happen henceforth and forever? Are the several "price indexes" used so sure-fire and fool-proof that implicit reliance may be placed on them?

Is the asserted average annual increase of about 3 per cent in world production and trade derived by use of a measure independent of the monetary one? And if so, what are the real grounds for the view that an equal increase must take place in total monetary stocks? Or is a 2.5 per cent increase, which some advocate, or a 2 per cent increase, which is held adequate by others, the figure to depend upon? Or is there some other figure?

Is not the oft-repeated necessity for distinction between secular and cyclic variations possibly an unconscious apology for the failure of the "quantity theory" to fit the facts since the war?

Is the general adoption of the gold standard anything more now than an experiment in progress, especially since the war, when so many of the financial transactions have been of a non-economic character?

Cannot international clearings be much further perfected?

Is it futile to expect a far more intelligent and effective coordination between the legislative, fiscal and economic units of nations than we now commonly see displayed?

If increase in the monetary supply of gold is so extremely vital, can there not be found some way of preventing the great dissipation (40 to 45 per cent or more of the total) into the arts and private hoards? Must governments continue to allow these uses to have first and unlimited call on the supply of a metal which in such large measure passes through governmental assay offices or refineries on the way from the mines to its ultimate destination? Is there not hope that by a wise campaign of education, hoarding as a non-earning investment may be caused to decline, not only for the future but as regards gold already absorbed?

If, as is so generally averred, gold reserves must bear some minimum ratio to world trade, what will happen as world trade continues to expand at a compounding rate while gold ores are progressively exhausted and at last yield a permanently declining output? Must not this question be faced in time? If as much gold has been absorbed in the last 25 years as in the preceding 400, what hope is there for holding to the pace very long?

Then why such general refusal to entertain alternatives in the thinking of the present? Are we not, in short, making ourselves slaves of an almost certainly unattainable doctrine or ideal?

Will the official value placed on gold have to be raised eventually? Will some form of bimetallism be reestablished, either with the more precious metals of the platinum group or with the less precious silver, or both? Or will the world accustom itself to progressively lower levels of general prices?

Whatever may be the proper answer to these questions, whatever may or may not be done in the way of economizing the supply of gold, whatever may be the eventual fate of the gold standard, those engaged in the mining of gold will continue, as in the past, in their effort to win from the earth every ounce of the metal that can be gained with profit. More cannot fairly be asked of them.

[General discussion on page 570.]

Possibilities of Gold from Low-grade Ore in South Africa

By F. Lynwood Garrison,* Philadelphia, Pa.

(New York Meeting, February, 1931)

The future of the Witwatersrand depends upon the possibility of mining and milling profitably the large tonnage of relatively low-grade gold ores known to exist in that area. The problem must be solved if the existence of the gold-mining industry in South Africa is to be saved and continue. To show that this disaster may be avoided, or at least postponed for another decade or longer, the following official statement is of interest: "There is a fairly large area of possible orebody on the East Rand of whose value and depth from the surface we have some though not very accurate knowledge. There is a far larger area south of the Central Rand which probably carries much payable ore, but at depths that at present appear too great to mine. Finally, there is a narrow strip of country over 100 miles in length stretching south from the Rand to Klerksdorp and beyond, which may contain large orebodies. The Witwatersrand rocks outcrop over portions of this ground and in places conglomerates occur in it that carry small bodies of payable ore. There is reason to believe that conglomerates of the Witwatersrand system exist over all this strip, though they are for the most part covered by later sediments. It seems probable also that there are large blocks of payable reef in this belt, at any rate in the area that lies close to the present western limit of the developed mines. The structure in this neighborhood is however very complicated and to prove the orebodies will be both difficult and costly."

Ores in the Witwatersrand

It should be explained in this connection that the conglomerates mentioned here were originally old gravel beds now consolidated into bedded sedimentary rocks, very hard because strongly cemented with secondary silica. The gold exists in this cement and is rarely found in the pebbles composing the rock or bed called locally reef. It is important to realize that these gold deposits are beds; not veins as generally understood by mining engineers and geologists.

In addition to these low-grade ore reserves we have also to consider the ore left behind in the old stopes of the upper levels near the surface,

* Mining Engineer.

which in former days of higher costs and less efficient equipment, for both mining and milling, was not profitable to mine. It seems probable that much of this ore may be actually richer than some now being mined at great depths. No record appears to have been kept of the probable tonnage of such ores, consequently the quantity and value are indefinite.

The great mountains of old tailings and refuse now dotting the landscape in the vicinity of Johannesburg appear to have been thoroughly leached and exhausted of their gold contents, having been in some instances rehandled and treated with cyanide twice or three times. Then again, we have to consider the debatable question, at any rate as applied to the Rand, of the decrease of value with depth. It was prognosticated as long ago as 1909 that, allowing for a normal decrease of gold contents with depth and no diminution in working costs, profitable mining could not continue below 5000 ft., whereas now the greatest depth attained is not far from 8000 ft., and while not unprofitable at this extreme limit operations certainly are increasingly expensive. Almost all mining districts begin with high values near the surface with relatively high costs to be followed later by necessarily lower grades and decreased costs. As working costs are reduced and the scale of operations extended, the menace of lower grade ore can be usually overcome and profits obtained. In the Rand, as has been intimated, there appears to be an almost indefinite amount of conglomerate carrying an appreciable amount of gold, and as working conditions or costs become more favorable the opportunities for mining low-grade ores become progressively encouraging and justified.

It might be added in this connection that recent prospecting on the gold-bearing beds which lie above and below the Main Reef series have shown that these slightly known deposits eventually may become important producers. Two West Rand mines are even now said to be profitably operating on the so-called "Bird Reef" of this section or series, and the results claimed to have been obtained at the "Consolidated Main Reef and Estates Ltd." are of such promising character that they may prolong the lives of the mines and possibly extend prospecting over a large area of new country. It is stated in a recent issue of the *Mining Magazine*[1] that this reef is 3000 ft. south of the Main Reef series, and strikes east and west over a considerable area. The annual report of this company claims that 2075 ft. of this reef was sampled, of which 1115 ft. was profitable at 5.9 dwt. over an average width of 51 in. It is also alleged that by June 30, 1929, the possibilities of this reef were definitely proved and an ore reserve of 287,210 tons having an average value of 5.0 dwt., over a width of 64.4 in., had been blocked out.

In addition to this development, the Randfontein Estates are opening what is known as the "East Reefs," which, judging from the history of

[1] (November, 1930) 287.

the district that had been mined previously by open-cast workings, ought, it is claimed, to contain large ore reserves. These reefs are about 1700 ft. east of the Main or Botha Reef series and strike more or less north and south for nearly seven miles. Although the value per ton of ore in them is relatively low, the beds are wide and occur at shallow depths, so that their operation should not be expensive. The East Rand area has also claimed some attention, but it seems that here only one of the series of these banket beds have proved of economic importance; namely, the Main Reef, the Main Reef Leader and the South Reef, the first being the most important.

Output and Costs on the Rand

On the Witwatersrand operating costs are based on the number of tons of gold-bearing rock milled and this average for the year 1928 on the whole Rand was in dollars about 4.74. Between the years 1910 to 1921–22 there was a progressive increase of from $4.22 to $6.16. From that date there was a decline to the end of 1926 and a slight increase since that date. This decline in costs was due to improvements in mining practice, especially in the drilling and blasting. On the other hand, costs are again mounting, owing to the great depths at which some of the mining is now being conducted. The present average is perhaps 5000 ft. vertical, increasing to an extreme of 7700 ft. in the Village Deep mine, which is the greatest depth to which mining has ever been attempted anywhere in the world.

In the year 1929 there was hoisted from the shafts of the operating mines of the district 37,600,000 tons of ore and rock. The amount of gold produced that year was valued approximately at $210,000,000. The working costs were reduced, from $6.15 in the year 1921, to $4.69 per ton milled in 1929. This was accomplished in the face of a small but steadily decreasing amount of gold in the ore. This decrease with depth is not necessarily the rule in all the mines of the district, but it is to be expected and is a common experience in metal mines all over the world. Considering the Witwatersrand as a whole, the average value of the ore milled appears to be in the neighborhood of from $5.50 to $6.00 per ton. Taking the lowest estimated working cost, which of course includes milling, at $4.69, and the highest average value of the ore, the margin of profit is but $1.31 per ton. These figures are only general, for the costs and profits vary in the different mines of the district and it speaks well for the management of some of them that there is any profit at all.

The total Rand output for the year 1930 was 10,719,760 oz., valued at £45,500,000 or approximately $218,400,000 which is the highest record ever attained. The mining dividends for 1930 were £8,644,309, an increase of £230,000 over 1929.

The gold bullion produced from the Rand contains an average of from 8 to 9 per cent silver, about 3 per cent of base metals, a few diamonds and a small quantity of osmiridium, which varies from 0.1 oz. or less to 1 oz. per 1000 tons of ore milled. Previous to 1918 no effort was made to save either the diamonds or the osmiridium, which in 1928 reached a value of $400,000 for the osmiridium and $530 for the diamonds.

Future Prospects on the Witwatersrand

Despite this creditable record of achievement and hopeful indications for the future, it should not be concluded that all is well with the Witwatersrand gold-mining industry. There is no denying the fact that some of its best mines of former days are practically exhausted of ore that at present is profitable; they cannot be expected to continue operations or even jog along with reduced production, and if completely abandoned are unlikely to be able to resume operations, for reasons well understood by mining engineers, even were conditions to become more favorable than those which obtain at present.

According to Dr. Hans Pirow, the Government mining engineer, in 1925 there were reserves of 320,000,000 tons of ore which may be expected to be mined and at the present rate of production this would last about 11 years, but there is necessarily a gradual shrinkage, which he estimated would be 13 per cent by 1935 in the gold output, by 1940 it would be 51 per cent and by 1945 it would amount to 78 per cent. He estimates the present reserves in the Rand and outside mines at 680,000,000 tons, representing a value of about $5,000,000,000, or somewhat less than the amount produced from the Witwatersrand in the 40 years since the district was opened. These figures seem rather confusing and perhaps contradictory, but the general opinion of those with whom I discussed the subject in South Africa was that the life of the existing mines would be about 15 years from 1930. This estimate of 15 years seems to me unduly pessimistic, although I must admit that the officials who made the estimate were in a much better position to know about existing conditions than anyone else.

There is no doubt that the industry labors under some very serious handicaps or disadvantages, which could be removed and probably will be ameliorated in the future. There is no doubt they exact a toll on the profits despite the high degree of technical and business skill now characteristic of this great industry. The prosperity of the industry is of vital importance to South Africa but it is of even more consequence to the world in general because of the steadily falling off of the world's gold production needed for its business, commerce and industry. It would be presumptuous of a visiting engineer like myself to discuss these disabilities, for they are mostly local and regulatory and could be modified or wholly removed, and doubtless will have to be if the vast reserves of

low-grade ore in this district are to be mined and made to produce to overcome the decreased output from the older mines.

The East Rand mines are a promising section of the Witwatersrand area. In 1929 they milled about 40 per cent of the total tonnage of the industry, returned 80 per cent of the total profits, and distributed 86 per cent of the total dividends. The other mines on the Rand earned only 20 per cent of the total profits and paid but 14 per cent of the total dividends, although handling 60 per cent of the total tonnage, produced 49 per cent of the whole gold output, spent 65 per cent of the total sum distributed in wages and 60 per cent of the amount expended in supplies, etc. In the same year (1929) there were 10 producing mines in the Rand district unable to pay dividends; yet these mines employed 7400 Europeans and 63,500 natives. Late reports from the East Rand area state that on the Geduld prospects point to the development of a large mine, and optimism seems justified by the developments in the Daggafontein section.

Rhodesia and the Transvaal

It is not to be assumed that because of the great volume and value of the Witwatersrand output of gold, or by reason of it, there has never been or there is not now any other gold-producing locality in Africa, more particularly in Rhodesia. Certainly there has never been and probably never will be another gold-producing area equal to the Rand in Africa or in any other country, but probably gold has been mined in Africa from the remotest antiquity; there are areas in the southern sections of this vast continent where there is today considerable gold production, and there is no reason to expect that it may not be largely increased in the future, although probably it will be insignificant as compared with the Witwatersrand. In considering the Rand, the fact should be borne in mind that it contains or contained in a relatively small compass a greater quantity of gold than was ever known to exist in a similar area anywhere in the world. It has, in fact, yielded in 40 years one-fourth as much gold as has been produced in the whole world since the discovery of America. And it is also well to remember that the great bulk of the gold produced in this long period came from easily worked placers or alluvial deposits wherein Nature had concentrated through a vast period of time the gold derived from the erosion of neighboring auriferous rocks. In other words, alluvial gold is the cream, easily skimmed off but never to be replaced within the ken of man. The cyanide process made it possible for one district alone on the Rand to produce in the year 1908 more gold than the total output of the world in 1888.

It has sometimes been assumed that the gold of Ophir mentioned in the Bible (I. Kings, 10) came from Africa, and attempts have been made

by several writers of romance, on no very substantial basis, to establish some connection in this with the ancient buildings, ruins and mines of southern Rhodesia. Mining engineers possessed of a flair for antiquarianism long have speculated as to the whereabouts of Ophir and whence came this enormous quantity of gold to Solomon, which according to I. Kings, 10–14, have amounted in one year to six hundred three score and six talents or about twenty-one million dollars reckoned in Babylonian equivalents. It is also said that Hiram in his navy brought from Ophir 420 talents (I. Kings, 9–28). We know that the string of ancient gold mines along the west coast of the Red Sea produced much of the gold of ancient Egypt; that they are characterized by narrow quartz veins carrying low values and have never paid when operated under modern conditions. It has been claimed, by some who advocate the ancient character of the old gold mines in southwestern Rhodesia, that Ophir may have been the modern port of Sofala on the east coast. This assumption, of course, has been predicated on the ancient origin of these mines and the associated ruined cities or structures which are an interesting and speculative feature of the hinterland back of Sofala. This has been ably disputed, although everyone you talk with in Rhodesia will assert their ancient character. Whatever may be the truth about this situation, it is a fact that very few of these old mines have proved of importance when operated under modern conditions. Some exceptions may be noted, as, for instance in the Cam and Motor, the Globe and Phoenix, the Shamva and the Lonely Reef, but the district as a whole holds out little promise of ever becoming an important source of gold supply. Of course one must bear in mind that probably all ancient mines were operated with slaves and that the value of gold in ancient times was immensely greater than at present. It may be that much if not most of this ancient gold was derived from easily worked alluvials now exhausted or covered up with desert sands or recent fluvial deposits. At any rate, Ophir remains an enigma variously placed in India, the East Indies, Abyssinia and Arabia along the southern coast, probably the most likely locality.

The question may now be asked, what have these speculations to do with out present problem to find gold for our modern needs? Obviously the answer is that if the ancients had mines that yielded such enormous wealth there must be something left in them which might be of great value when operated by modern methods.

In the Transvaal, aside from the Witwatersrand, the outlook for gold production is much the same as in Rhodesia. Although gold has been mined in the Transvaal since 1868 the total yield has been only $75,000,000, which is rather insignificant as compared with the five billions and over of dollars worth produced on the Rand in 40-odd years. Some authorities estimate that the Witwatersrand may be good for one-third

as much more before ultimate exhaustion, independent, I believe, of what might be expected from the low-grade ores which cannot be profitably mined at present.

It would seem, therefore, as far as one can judge, that the future mining industries of both the Transvaal and Rhodesia will be chiefly confined to the other minerals with which these large countries are abundantly provided. Very little alluvial gold has been found in the Transvaal. As a rule the outcrops on the Rand have not been accompanied by nuggets, gold dust or specimens of fine gold. The ore itself is rather unattractive and seldom shows visible gold, yet it has long been the greatest gold field ever discovered.

Lomagunda and the Gold Coast

There are at least two localities in Africa where gold-bearing conglomerates have been discovered—in the Lomagunda district along the Hunyani river northwest of Bulawayo and in the Gold Coast colony at Tarkwa. It was fondly hoped at one time that these deposits might prove another Witwatersrand, but they have both failed to be of distinctive importance. Although the Lomagunda deposits resemble conglomerates, it has been claimed that they were not real sediments but eruptive rocks more or less rounded into breccia when in a semiplastic condition. On the other hand, according to Gregory[2] they are lenticular masses of conglomerate interbedded with quartzites that were deposited as beds of sand. The pebbles themselves are eruptive rock. Whatever they are, it seems more than likely they will never be of much economic importance.

In the Gold Coast colony gold appears to be widely distributed but in only a few places is it sufficiently concentrated to be important. The conglomerate or "banket" mines are situated in the Tarkwa district, which is 45 miles from the coast at Sekondi; it is claimed they average from $8 to $10 per ton in value. Despite its name, the Gold Coast does not appear to hold out much hope of becoming an important gold-producing country; but of course it should be remembered that much of this area is probably largely unexplored by geologists and is a difficult country to prospect. At the same time, it should be pointed out that this colony since 1901 has yielded from its mines something over 6,000,000 oz. of gold.

The copper ores in the Belgian Congo sometimes carry small amounts of gold, but the copper ores in Northern Rhodesia, which are of vast extent, appear to contain practically none at all, and although there may be exceptions, it seems improbable that any of these copper deposits, as far as we now know, will ever be a notable source of gold supply. Some

[2] J. W. Gregory: The Ancient Auriferous Conglomerates of Southern Rhodesia. *Trans.* Inst. Min. and Met. (1905–1906) **15**, 563.

seemingly important gold discoveries have been made elsewhere in the vast extent of the Belgian Congo, but it is difficult to obtain reliable information about them because the Belgian colonial authorities seem not to welcome publicity or visiting engineers and geologists.

Africa appears to be conspicuously lacking in alluvial gold deposits of notable extent. The ancients may have operated them in Southern Rhodesia; if so they are evidently exhausted, as they have been in almost every other part of the world. At any rate, it seems unlikely that they were even large or extensive. Buried stream channels, known as "deep leads" in Australia and California, which are simply old river beds covered by comparatively recent basaltic flows or erosion products from regional drainage, appear to be unknown in southern Africa. In some instances their exploitation has been profitable in Australia, although less often in California. In Australia it has been claimed that this failure was due to a neglect to study the laws governing the flow of water in the old and existing drainage.

Outlook for Future

On the whole, the outlook for future gold discoveries in South Africa and in Rhodesia is not good and excepting the extensive low-grade deposits on the Witwatersrand, the gold ore reserves of that vast territory appear to be very limited. I am disposed to think, however, that this broad statement should be qualified, in view of the fact that geological exploration and prospecting has not yet covered and probably cannot for some years to come overspread such a large country. The outcrops on the Rand were never very promising and had they been obscured more completely with detrital material might long have remained unsuspected and undiscovered. So it would seem unwise to draw any sweeping conclusions from our present knowledge of the country, which is of profound and ingratiating interest to geologists as well as to every student of natural science. The fact that the safety of life, health and comfortable living is to be found almost everywhere in all countries under British rule is a strong factor favoring the continued rapid development of Africa. With the exception of Russia no other part of the world seems to have potential mineral resources equal or superior to those of central and southern Africa. Oil appears to be about the only important mineral lacking but there is no reason at present to suppose that it may not be found eventually. Russia, or Russia and Siberia, undoubtedly contain important deposits of gold largely unworked and undeveloped. In view of the hostile and intransigent attitude of the present Russian Government there is little hope for safe foreign investments in that enormous country, destined inevitably, with its great virile, white population, to become a dominant factor in the world's industries.

[General discussion on page 570.]

The Gold Situation

By George E. Roberts,* New York, N. Y.

(New York Meeting, February, 1931)

Abstract

The maintenance of the common gold standard is the most important cooperative undertaking in the world. The war broke up this relationship and brought about a state of great disorder in the currencies and exchanges. After the war the countries all began to grope their way back to the gold standard as the best means of establishing trade and financial relations with each other. With this movement began a general decline in commodity prices. The question has been raised whether the world's return to the gold standard was not forcing prices back to the prewar level. Also, the question is raised whether the world has not outgrown the supply of gold as the basis of credit.

Because Europe readjusted its monetary units to conform to the higher level of prices there is no pressure on prices on this account. Furthermore, a large amount of gold formerly in hand-to-hand circulation passed out of circulation and has been added to the central bank reserves. The gold reserves of 36 countries increased approximately 100 per cent from the outbreak of the war to June 30, 1930. The increase is much larger than the average increase of prices in any of the important countries, at any time since 1925. As gold in the central banks of issue is far more effective for all business purposes than gold in circulation, the gold scarcity argument has little basis as applied to this time. The abnormal distribution due to the war was to the disadvantage of business everywhere. Credit was scarcer and dearer in some countries. The inflation in this country might have been greater had the production of gold been greater.

The argument supporting the theory of a scarcity of gold assumes as fundamental that general price changes must be accounted for by changes in the price equation on the side of money. This is disproved in every period of business depression. A downward price movement is characteristic of every period of business recession.

Important discoveries of gold must be made even to maintain present gold production, in the face of which the world's business is bound to continue increasing. It cannot be doubted that some way will be found

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to handle an increasing volume of trade and business whether gold production keeps pace or not, but it is of great advantage to have the world united upon a common standard of value. If all payments were made by paper instruments which could be promptly brought together, they would practically offset and cancel each other with very little demand upon the reserve holdings. That, in fact, is the tendency.

It is now the rule to concentrate the final reserves of all countries in central banks, and those banks supply currency as required. Gold is no longer required for any monetary use, except as the standard of value and for the settlement of balances, and the last statement may be limited to international balances. There is no movement of gold from one place to another—the system is simply a matter of bookkeeping. The gold in the Gold Settlement Fund is not even segregated from other gold in the United States Treasury. The constitution of the Bank of International Settlements suggests yet more intimate and more cooperative relation between the banking systems of all countries. As the world wealth increases, international securities will be another factor in maintaining the world equilibrium.

There is need for a more general understanding that no country has anything to gain by piling up claims against another country which must be settled in gold. It is to the interest of all peoples that the normal equilibrium in trade and capital movements shall be maintained. It is not necessary to pile up gold reserves indefinitely. The world is facing no crisis on account of gold; there need be no crisis on that account even if gold production shall show a declining tendency.

With the use of gold only for the settlement of balances, new possibilities of credit expansion appear and management looms up as of greater importance than before. If there must be management, there is still the importance of maintaining an international standard, and of securing international cooperation in maintaining its stability. War may destroy the entire system, but it would seem to be about time that all peoples fairly grasped the fact that there is no place in modern, highly organized society for war.

[General discussion on page 510.]

Sources and Trends in Gold Production

By R. J. Grant* and John B. Knaebel, Washington, D. C.

(New York Meeting, February, 1931)

Abstract

This paper outlines the trends in gold production since the discovery of America, in the world as a whole, and in the principal producing regions as well. World production climbed at an average rate of about 20,000,000 oz. per 5-year period until the record all-time output of 111,307,000 oz. in the period ending with 1915. Production fell off during the war period, but has been mounting since 1925, and will probably be nearly 100,000,000 oz. for the 5-year period ending in 1930. Total world production of gold from 1492 to the end of 1930, as closely as can be determined, is more than 1,061,000,000 fine ounces. More than one-half of this total, or about 575,000,000 oz., has been mined since 1900.

In percentage of world production, the United States jumped from 4.8 per cent for the decade 1831–1840 to 44.5 per cent for the five years ending with 1855. This is the highest figure ever reached by this country, although, with the exception of the 10-year period ending 1865, it ranked first among the nationa from 1849 until the end of 1905. During the 5-year period ending with that year, the United States produced 24.7 per cent of the world's gold. The tremendous increase from South Africa following 1890 resulted in that country's becoming the largest contributor for the five years 1906–1910, a position it has held with increasing dominance ever since.

Relegated to second place in the period 1906 to 1910, the United States produced nearly 20 per cent of the world's new gold until 1920. From 1921 to 1925 it averaged 13.6 per cent and since then it has continued to decline in relative importance, accounting for only 10.9 per cent in 1927.

Final figures for 1930 may place Canada second among the gold producers of the world.

[General discussion on page 570.]

* Director of the Mint.

Gold Supply in its Relation to Currencies and World Commerce

By A. C. Milner,* Salt Lake City, Utah

(February Meeting, February, 1931)

Abstract

In a gold standard world which possesses insufficient metal to meet the every-day needs of all, confidence becomes the vital factor, the very foundation upon which the machinery for the expansion of gold into currency, and credit, rests. The gold supply, affected by many influences, moves constantly from one point to another, but not always to that point where it is most needed. In the period 1928–1930, the combined gold holdings of the United States and France increased $1,370,936,000, and this with new gold adding only $600,000,000 to the monetary gold stock. Therefore, the gold stocks of other countries must have been depleted during the period by $770,936,000. This uncontrolled shifting throws the delicately adjusted instrumentalities of both credit and commerce out of equilibrium.

If a given nation's gold stock shifts so as to bring it substantially below the legal reserve ratio set up against it for currency issues, the effect can easily extend to the currency issues and banking credit volume of one or more nations which may have based their gold exchange standards upon the gold stock of such nation. Thus the substitution of paper based on gold in place of the actual gold is an inflation of the currency, which carries through into an uncontrolled pyramiding upon the entire machinery of credit based on gold.

In 1929 Europe, South America, Africa and Asia had overissued currency based on gold. America, Oceanica and Japan, which have not overissued their gold currencies, would in normal times require their entire present gold stocks and, as Continental Asia is largely upon a silver credit basis, from a domestic standpoint, it would appear that the gold problem is not so much one of maldistribution, as it is one of inadequate supply.

It would appear that some method of increasing greatly the velocity of gold, within the safe reserve point, should be devised, or that silver, of which estimates indicate that 7,000,000,000 oz. are available for

* Milner Corporation.

monetary purposes, should through international cooperation and agreement be placed upon such basis of value, in ratio to gold, as will relieve the pressure upon gold, as related to the domestic credit requirements of Asiatic peoples, thus restoring their commodity price levels, and potential purchasing power, to the great advantage of the gold standard nations, and their producers, wage earners and exporters.

DISCUSSION

(Albert O. Hayes presiding)

J. Boyd, Los Angeles, Calif.—Having mined gold in South Rhodesia and Australia, perhaps a word from me about the possibility of increased gold content in the ores at depths in these mining fields might dispel the air of gloom about the present gold situation.*

In South Rhodesia gold occurs very close to the surface and often pinches out at very shallow depths. The reason for the gold-mining boom in Rhodesia in 1900 was the discovery of ancient workings, which went down to depths of 100 to 150 ft. Numerous mining propositions were floated on the London market in the early part of the century merely on these discoveries on ancient workings. Although most of the veins, unfortunately, petered out in depth, I believe there is still a chance in South Rhodesia for more permanent gold bodies to be found.

I have recently had correspondence with some of the executives of the Chamber of Mines in western Australia, and am informed that the gold subsidy of one pound per ounce is not only to stimulate prospecting but also to make possible the working of gold orebodies of lower grade. Many orebodies which contain gold to the value of from $4 to $5 per ton can be worked at a profit when aided by the subsidy mentioned.

I agree with one of the speakers that a great deal of gold will come from the deeper seated orebodies. Just before the war I was mining to a depth of 2300 ft. on the Chaffers mine in the Kalgoorlie gold field. There were indications that the sulfotelluride ore would increase in value at a depth of about 3000 ft. In the past year this has been proved a fact, and this mine, now being operated by the Lake View Star Co., is showing wonderful ore at about the horizon mentioned. There is better ore in this mine at this horizon than there was in the upper levels. I think it is quite possible, in fact probable, that the ore will increase in value in the Kalgoorlie field to depths as far as we can mine; *i. e.*, 5000 to 8000 feet.

R. D. Hoffman, New York, N. Y.—Surface examinations of prospects are, as a rule, incomplete and inconclusive. Possibilities of errors in judgment are great when we consider how small is the proportion of gold to rock gangue in ore of excellent commercial grade. Also, the character of gold occurrence is as various as the different camps in the world, often changing within the confines of a single camp. A third factor not generally considered is the large part played by individuals in the development of gold mines, particularly in Cripple Creek, Colo.; Goldfield, Nev.; Juneau, Alaska; Kirkland Lake, Ont.; and now Quebec. On the whole I would say that all predictions are tentative, are subject to drastic change, and should be frankly offered as such.

As regards Canada, I was fortunate enough to witness a period in which Kirkland Lake blossomed from what many geologists and engineers termed "an interesting occurrence of gold" into a camp now producing at about a yearly rate of $18,000,000. One or two men persisted in developing a few comparatively small oreshoots, and their

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* February, 1931.

efforts were crowned with phenomenal success. In 1921, any geologist who predicted an $18,000,000 camp would have been termed crazy. We had no basis for this prediction, yet time and work brought it out. Similarly, in Quebec, little was known at first as to future possibilities. I prospected there in 1921 and 1922, before Noranda was developed. Like the rest, I was searching for gold, but after Noranda came in the emphasis turned to copper. Some gold developments continued in the sediments to the south and this was mainly inconclusive. On one property several hundred thousand dollars was expended. Then the owner added $300,000 of his own and succeeded in developing what we consider a good mine, the Granada, with a present annual production of approximately $600,000. This in itself is not startling, but it opens up the whole southern section of the Rouyn area, hitherto considered disappointing.

A few years ago few of us would have been willing to spend money in a country like Canada, where the overburden, water and paucity of outcrops makes preliminary exploration work costly and inconclusive. Now, however, with lower costs, encouraging developments in the gold camps, new discoveries, added incentive, the picture is radically changed. I may be a bit optimistic, but I believe I am safe in predicting more gold camps in Canada. These, however, will be found over a long period of years.

I believe that the chief function of geologists will be to render valuable work by outlining the so-called favorable areas for development and eliminating the areas that are definitely unpromising and unproductive.

A. C. Daman, Denver, Colo.—The speaker referring to gold was smiling, but with silver quoted at 26¢ and a fraction, I cannot smile. Today we are concerned regarding the condition of a silver patient, not a gold patient.

A generation ago discussions on monetary relations were rife, with the parity existing between silver and gold, and a ratio of sixteen to one was proclaimed, which was close to the truth.

Statistics of the United States Treasury Department show that the total production in fine ounces of gold and silver *throughout the world*, from 1493 to 1929, inclusive, was in the ratio of 1 oz. of gold to 14.15 oz. of silver. The total production in fine ounces of gold and silver *in the United States* from 1792 to 1929, inclusive, was in the ratio of 1 oz. of gold to 14.22 oz. of silver. The close agreement between these ratios as compared with the present ratio of 1 to 80 compels our profound consideration.

A powerful advocate of bimetallism was Gen. Francis A. Walker, superintendent of the Ninth and Tenth Censuses and president of the Massachusetts Institute of Technology. His works on The Science of Wealth, Political Economy, Money, Wages and Rent are classics. He presents the situation of gold-using countries having vast accumulations of wealth derived from the industry of the past—their productive power is large, wages are high; in them trade and industry are organized with a great degree of complexity and minuteness. On the other hand there are silver-using countries, embracing an aggregate number of inhabitants many times greater than those of the gold-using countries. In these silver-using countries the facts of industry and the habits of the people, in respect to exchange, are such as to make gold an impossible money. In such countries as China and India, where practical convenience, sentiment and habit give a decided preference to silver, it is not reasonable to anticipate that these countries will soon, if ever, pass over from the silver-using to the gold-using group.

Walker submits two considerations of the bimetallist which he alleges to be of vast importance to the world's trade and industry. The first is the establishment of a par of exchange between silver-using and gold-using countries. The second benefit is that the two metals, thus bound together, would constitute a better money than either metal by itself. Inequalities of production would tend to equalize each other, resulting in greater uniformity in the production of the compound mass, and hence a greater steadiness in the value of money.

With the commerce and industry of the world increasing, while the production of gold is rapidly decreasing, does it not seem that we must resort to the use of silver to stabilize values and promote trade between the silver-using and gold-using countries? A plan recently advocated by our organization* and submitted to our friends throughout the world, involves the following points:

1. A more general distribution of metallic coins, not only in India and China, but also in this country. Prevailing demand in this country is for coins of small denominations. Greater use of silver throughout the world will increase the demand for coinage purposes, thus increasing the price of silver.

2. The use of an international dollar as a medium of exchange throughout the world. Suggestions have been made of an international dollar on a gold reserve basis, part gold and part silver, or entirely made of silver, with a perforation through the center to distinguish such coin from our present half dollar. This smaller coin would eliminate the common objection to our present heavier silver dollar.

3. Establishing a temporary bonus on the production of gold. Already a bonus has been adopted in Australia and no doubt will result in increased production of gold in that field. The price of $20.67 per ounce of gold was fixed in 1844, and comparing the production costs of labor, materials and supplies of that period with the costs of today, offers an argument for readjustment of a fixed price for gold.

4. Cooperation between government agencies and producers of gold and silver, whereby ores of lower grade may be utilized through the use of improved equipment, better metallurgical methods and business organization, thus enabling the smaller mines to operate successfully.

These four factors embody a number of the opinions that have been advanced within recent months. Although applying directly to the mining industry, these points will have a direct and permanent effect upon every individual, both national and international: For when we sum it up, is not mining one of the basic industries?

H. Emerson, New York, N. Y.—I do not speak without reflection and first-hand experience when I say that there is no reason except cowardice why we cannot mitigate unemployment by increasing the price of silver slowly and steadily, by a single signature to a simple legislative act in Congress. The fall in silver would then cease to be the cause of distress that it is all over the world today; it would become an influence of amelioration. I have just returned from twice around the world in the last two years. The United States should advance silver to a ratio of at least ten to one.

Recently I discussed the subject with one of the Chinese bankers. He was asked what the value of silver would be if it were wholly demonetized. He said that there is no reason why silver should not drop lower than the price of copper. The great use of silver is as an alternate money metal. We all know that silver is better as an electrical conductor than copper, but everybody would be afraid to substitute the use of silver for copper for fear there would not be enough of it. If silver is dismissed as a money metal its value may drop until it becomes an alternate for copper.

On the other hand, if England, which has been primarily responsible for silver degredation, should so change her policy in India as to make silver rise again, would we welcome it or not? If silver once more should go up to $1.33 an ounce, as it did only a few years ago, would it be an advantage to the world or not? That is the question. Would it be a disaster to have silver go down to the price of copper? Would it be an advantage to the world if silver should rise again to a stable alternate value?

Why should we, the greatest nation on earth, with the greatest power, wait for any other nation on earth when it lies in our hands to stabilize the price at any rates most to our advantage, that of the world as well? Why should we wait to confer with

* Denver Equipment Co.

England, since England controls 70 per cent of all the gold output of the world? Why, when up and down the Pacific Coast from Bering Strait to the Straits of Magellan the silver mines of the world lie unworked? Why, when Mexico, to the south of us, is in the depths of despair on account of the shutting down of her silver mines? When Japan and China and India with their 800,000,000 inhabitants cannot buy from us, because our prices are too high for their silver wage scale?

I was in Shanghai last February when silver dropped 33 per cent and I saw what happened. Why should we not in this country take the step that would be of incalculable advantage to our industries, to our unemployed, to our production, and also to the world at large?

Mr. Roberts admits that gold is decreasing. He claims we should have one standard. I have no word to say against the gold standard. I would be the first to strengthen it. I am entirely willing to make it even more the standard than it is today, but it is one of the greatest fallacies on earth to have a single standard of anything. That is not nature's way. There are always alternates in all fundamentals. If there is no alternate, there is a monopoly, and distress begins.

Take life. There are two ways of propagating life on earth, the method of budding—the old trees in California that have lived for 3000 and 4000 years, budding, budding, budding always. Then there is the other alternate method that comes to supplement it, that of conjugation. One method perpetuates, the other selects and improves.

What would we do if we only had one building material? Or one kind of food or one kind of clothing? When there is only one, no alternate for stabilization, the fluctuations are far more severe. There are 100 times as many hens' eggs used in the United States as ducks' eggs, and yet they command exactly the same price because they are alternates. Alternate commodities, irrespective of the fluctuation in the quantity of their production, do not change in relative price.

The ratio of silver and gold has always been determined by the strongest commercial nation. It has never rested on the relative production. It has rested on the ratio established by the strongest commercial nation. Before the discovery of America that ratio was determined by the Portuguese and after America's discovery it was determined by Spain down to the end of the eighteenth century. The reason we adopted the ratio of 15 to 1 was because it was the Spanish ratio and had been for many years. The French, having conquered Spain, adopted the ratio of 15½ to 1. The United States was weak financially and commercially. Its ratio of 15 to 1 was disturbed. All American gold was exported. Instead of changing to 15½ to 1 we changed to 16 to 1 and lost all our silver.

The mistake in this question of gold and silver lies in the idea of a fixed ratio. We do not want anything that is fixed. We want dynamic changes, adjusted to the changing conditions of the world.

Our government would not have to buy a single ounce of silver. There would not be an ounce of silver deposited in our mints. All our government would have to do would be to say, "We will begin with silver at 20¢ an ounce, if you choose, or 25, and anybody who deposits silver at the mint will get a silver certificate in exchange, receivable for taxes. On the first of March it will be 25¢; on the second day of March, it will be a little more; and each day thereafter it will follow a rising curve." And who would deposit an ounce of silver when it was going up in value? Nobody. Silver would inevitably come up.

What is the production of silver compared to the $5,000,000,000 of taxes this country buries every year? What is the small production of silver compared to the enormous needs of the world? All we need in this matter is to stand as Americans and not wait for any other nation on earth to come and tell us what to do to stop our own troubles. We do not do it with anything else. If we have an epidemic, we

adopt our own methods of meeting it. I am not in favor of waiting for conferences when we alone as other nations have done in the past, can do what is necessary for our own people.

I am not speaking as a silver miner, because I never had any interest in silver; I am talking because we have 8,000,000 people unemployed in this country and it could be stopped in three-months' time.

V. L. HAVENS, New York, N. Y.—I am pleased with this general note of pessimism. I think true leadership is generally found among those who have sufficient imagination to recognize dangerous trends and also have sufficient courage and ability to face and overcome those dangers. It is only through such notes of pessimism that we discover the trends. They are in no sense definite prophecies. They are merely a spur to solve the problem which, of course, can be solved.

Contents of 1931 Volumes

Abstracts

On the following pages are abstracts of papers published by the Institute during the year 1931 as *Technical Publications*, *Preprints*, and in bound volumes. For abstracts of papers that appear in bound volumes in 1931 but that were published as *Technical Publications* in 1930, see the Transactions, 1930. Papers that appear in the volume in which this list is printed are not abstracted here.

Many of the *Technical Publications* have been reprinted in bound volumes. Information regarding this disposition, and number of pages in each paper, may be found in the list beginning on page 616.

The abstracts are grouped as follows:

Metal Mining

Use of Models for the Study of Mining Problems. By Philip B. Bucky. (*Tech. Pub. No.* 425, 12,000 *words.*)—This paper describes a research project undertaken to provide a means of studying the behavior of weighty structures such as mines, dams, foundations, etc., with models of the same material in the laboratory. The paper shows that in general it can be done. The underlying theory is explained, the apparatus which was developed and built to do this is described, and the results of laboratory experiments with sandstone mine structures are given. There is discussion of the accuracy of results with the present equipment, the model size necessary, and of the size and type of equipment necessary for the obtaining of results of practical value in the field.

Milling and Concentration

Ball Mill Studies, I. By A. W. Fahrenwald and Harold E. Lee. (*Tech. Pub. No.* 375. 25,000 *words.*)—This paper presents the results of an extensive experimental study of some of the numerous elements entering into ball mill grinding. All apparatus used was equipped with ball bearings. Every experiment was accompanied by a power measurement accurate to 0.005 hp. Quartz was used. Each element studied, such as ball load, was varied systematically. Power curves were obtained for the mill for three general sets of conditions; namely, (1) balls only (2) balls and quartz and (3) balls, quartz and water. For each general condition, power curves were obtained for various mill speeds, ball loads and ball sizes. For conditions 2 and 3, feed size and weight and pulp dilution also were studied as variables. The mill output, in grams of finished product per unit of time, was obtained for each test. In a general way, output (mill capacity) paralleled the power data and this attached

much interest to optimum points on the power curves. The work logically led to a study of mill speeds above critical and the results in this range, which were in a way predicted from our mathematical analyses, rewarded us in a large way for our efforts. A mill speed of the order of 140 per cent critical, which is more than double that employed in present practice, and a ball load in the neighborhood of 20 per cent mill volume, which is about one-half that employed in present practice, gave more than 100 per cent increased mill capacity over the standard conditions now used. Viscosity, angle of nip and circulating load also were considered. The work points clearly to the fact that for any given mill there are optimum conditions for maximum output and that this condition is remote from those employed in present practice.

Ball Mill Studies, II.—Thermal Determinations of Ball Mill Efficiency. By A. W. Fahrenwald, G. W. Hammar, Harold E. Lee and W. W. Staley. (*Tech. Pub. No.* 416. 6000 *words.*)—The published calculations of several investigators give very low absolute efficiency values for ball mill comminution. These figures vary from less than one per cent to several per cent. All of these calculations are based on the use of a surface energy value for the material in question and on total surface produced in the reduction operation. Both of these data are of doubtful reliability. This paper presents a method of determinating relative ball mill efficiency, based on thermal measurements, that involves neither of these data. It is simple and direct. By this method, much higher relative efficiency values for laboratory ball mill batch grinding were obtained. Under a new set of ball mill conditions, as high as 25 per cent efficiency was recorded. Efficiency determinations based on thermal measurements also show the large improvement in the grinding capacity of a given mill when operated above critical speed and with small ball loads.

Experimental Flotation of Oxidized Silver Ores. By H. S. Gieser. (*Tech. Pub. No.* 401. 4000 *words.*)—An attempt was made to find some reagent that would help decrease the losses incurred in floating an oxidized silver ore along with straight sulfide material. A few metallic salts were tested and it was found that salts of the nature of sodium aluminate have some accelerating action. Quite a number of organic-sulfur and organic-sulfur-nitrogen compounds were tested and from a study of the data the latter appeared more promising. The reaction product of cresylic acid, sulfur, and an amine such as anilin or ortho-toluidine gave a slightly higher recovery with a higher grade product than other compounds tested. Grade of product is important because of location of the mine.

Flotation of Minor Gold in Large-scale Copper Concentrators. By Edmund S. Leaver and Jesse A. Woolf. (*Tech. Pub. No.* 410. 15,000 *words.*)—Considering the large tonnage of copper ores that are now concentrated by flotation, as computed in a recent review, the gross gold content is surprisingly large, probably amounting to over ten million dollars per year in the United States. The amount of gold that occurs in the milling ore from most of the large scale copper mines is so small (less than 50 cents per ton) that no unusual care and very little or no additional expenditure per ton of ore are necessary in the recovery of this gold.

Lime is generally used as a depressant for pyrite during flotation for the recovery of copper sulfide minerals, resulting in higher grade copper concentrates with only a slight loss in the percentage of copper recovered. The authors have shown that high lime or solution high in calcium ions is also a depressant for gold. Their experiments on several well-known ores show that by careful regulation of the alkalinity of the mill solutions, a higher flotation recovery of the gold can be obtained together with equally as good copper recovery in a good grade of copper concentrates. If the solutions are near saturation with calcium salts, fully 50 per cent of the gold may be depressed and lost in the tailing. A pH range of 7.5 to 9.0, due to lime, offered the best condition for the flotation of the copper, silver, and gold. An alkalinity of 10 pH, or above, very much lowered the recovery of the gold. It is

important for the recovery of gold that the solutions in the flotation circuit be analyzed for calcium ions at regular periods and at each step in the flotation practice. The lime should be added when and where needed and not in excessive amounts at the head of the mill.

Effect of Particle Size on Flotation. By A. M. Gaudin, J. O. Groh and H. B. Henderson. (*Tech. Pub. No.* 414. 11,000 *words.*)—A study of the effect of particle size on flotation was made by two methods. One method consisted in grinding, floating, sizing, and analyzing mineral mixtures. The other method consisted in sizing and analyzing the products of several mills. Sizing was done by elutriation in water and acetone to as fine as 2200 mesh (about 5 microns). Broadly speaking, the following conclusions were reached: (1) results obtained by the two methods of study are in agreement; (2) the usual notion about the relative ease of flotation of very fine particles does not agree with the facts; (3) there is a time sequence of flotation with regard to particle size, the medium-coarse particles floating first; (4) no successful method of floating extremely fine pulps has been found; (5) recovery is optimum in a certain well-defined size range; (6) selection is optimum in another well-defined size range. Several hypotheses are discussed to account for the nonrecovery of fine, otherwise floatable particles.

Iron and Steel

The Open-hearth Steel Process as a Problem in Chemical Kinetics. By Eric R. Jette, New York, N. Y. (*Tech. Pub. No.* 380; also *Trans., Iron and Steel Div.,* 1931. 16,000 *words.*)—The possibility of developing, on a theoretical basis, an equation for the rate of carbon elimination in the open-hearth process is discussed. It is pointed out that until the mechanism (*i.e.,* the slowest chemical reaction) involved in the oxidation of carbon is known by direct experiment, theoretically derived equations must be considered as speculative. On the assumptions that the rate-governing reaction is $FeO + C \leftrightarrows Fe + CO$ and that one may, therefore, consider the open-hearth process as a group of individual processes each of which affects the concentration of FeO dissolved in the metal, the author outlines a method of developing an equation for the open-hearth process. It is emphasized that in a complex system such as the open-hearth, the accuracy of the final equation will depend on (1) the accuracy of the description of the process as a whole, (2) the accuracy of the equations for the individual processes, and (3) the correctness of the way in which the individual equations are combined. The difficulties in the way of obtaining a general equation are outlined, several of the individual processes discussed in some detail and, in particular, it is shown that in the degree to which the diffusion of FeO from slag to metal is important in determining the rate of carbon elimination, there will always be uncertainty due to the impossibility of defining and measuring certain physical and mechanical conditions existing in actual practice.

A Thermodynamic Study of the Phasial Equilibria in the System Iron-carbon. By Yap, Chu-Phay, New York, N. Y. (*Tech. Pub. No.* 381. 3000 *words.*)—This is the first of a series of theoretical papers on the iron-carbon system. A simple critical analysis of the ordinary constitution diagram of the iron-carbon system yields valuable information regarding the nature of the equilibria in the liquid and solid states. It is shown, for example, that the solute in the melt is Fe_3C, but in solid γFe (austenite) the solute is carbon. The thermodynamic laws of the depression in the freezing point of solid solutions have been developed and discussed. Using the usual method of plotting the logarithm value of the concentration of the solvent (in this case, the ratio of concentrations of the solvent in the liquid and solid phases) against $1/T$, curves are obtained. A rigorous thermodynamic analysis of these curves has been

shown to yield considerable information regarding the internal equilibria of the system, with which the phase-rule does not deal altogether. In this way, it is shown again that the solute in the melt is Fe_3C, but in the solid γFe the solute is carbon. Various heats of reaction (fusion and transition) and other thermal data have been derived and discussed with a view to correlating the conflicting experimental data obtained by different investigators of the system.

Influence of Dissolved Carbide on the Equilibria of the System Iron-carbon. By YAP, CHU-PHAY, New York, N. Y. (*Tech. Pub. No.* 382. 6000 *words.*)—This is the second of a series of theoretical papers on the iron-carbon system. The solidus lines obtained as a result of the different methods of investigation fall naturally into three classes; namely, (1) a straight solidus originally suggested by Roozeboom and supported in the main by the work of Carpenter and Keeling, Andrew and Binnie, et al., using the thermal analysis method; (2) a slightly curved solidus obtained by Honda and his associates, using the magnetic and resistance methods; and (3) Gutowsky's solidus obtained by the Heycock and Neville method of quenching at successively higher temperatures. Although the thermodynamic method as shown in the first paper indicates that the straight solidus line is correct, no explanation has ever been advanced to account adequately for the disagreement in the shape and position of the solidus lines obtained. The nature of the solute in austenite is theoretically discussed. On the basis of such discussions, the data obtained by Honda and Endo on the change of magnetic susceptibility of steels with respect to temperature was critically subjected to a graphical analysis, which is shown to yield a rational explanation of the curved solidus lines.

Viscosity of Blast-furnace Slags. By RICHARD S. MCCAFFERY, Madison, Wis. (*Tech. Pub. No.* 383. 20,000 *words.*)—These three papers give a detailed description of the apparatus and method for determining the relations between the viscosity, temperature and composition of the system silica-alumina-lime-magnesia, within which the blast-furnace slags are found. Typical test log and data sheets illustrate the method of calculation. The reproducibility of results is emphasized, and compared with results of other workers. Magnesia is shown to have more influence than lime in reducing slag viscosity. For detailed use in blast-furnace operation or in other investigations, results are summarized in numerous diagrams and tables. A method is given by which any particular slag can be located on the diagrams and the effect of changes of composition and temperature can be determined readily and used to secure greater regularity of blast-furnace operation.

A Statistical Analysis of Blast-furnace Data. By RICHARD S. MCCAFFERY and RONALD G. STEPHENSON, Madison, Wis. (*Tech. Pub. No.* 384; also *Trans., Iron and Steel Div.*, 1931. 4000 *words.*)—This paper outlines a method of analysis of daily operating reports of a blast furnace, based on the use of mechanical accounting machine methods. It indicates the suitability of the method for an extended statistical analysis of such data. The effect of variation of slag composition in blast-furnace practice, on economically important factors such as coke consumption, tonnage and sulfur elimination is shown, and a basis indicated for correlating the results obtained in commercial practice with those predicted from a study of the quaternary system silica-alumina-lime-magnesia.

Air Discharge of Circular Tuyeres. By RICHARD S. MCCAFFERY and DANIEL E. KRAUSE, Madison, Wis. (*Tech. Pub. No.* 385; also *Trans., Iron and Steel Div.*, 1931. 3300 *words.*)—This paper describes a laboratory method of testing the efficiency of blast-furnace tuyeres with a view to developing improved design. It includes a short discussion of the conditions in a modern blast furnace and the possibilities of improving these conditions by using a different type of tuyere. The shapes of the air streams from various types of tuyeres were determined and some of these shapes are illustrated, together with a description of the apparatus used.

Bright Annealing of Steels in Hydrogen. By FLOYD C. KELLEY. (*Tech. Pub. No.* 338; also *Trans., Iron and Steel Div.*, 1931. 4000 *words.*)—This paper deals with the causes of oxidation of stainless iron and low-carbon steel in hydrogen furnaces and methods of preventing it. The fundamental knowledge obtained from a series of experiments which are described is applied to the continuous annealing of these materials. The chief causes of oxidation are water vapor held by the furnace insulation, moisture condensed in the cooling chamber, and air which diffuses into the furnace against the pressure of the hydrogen. Low-carbon steel has a critical temperature range of oxidation which does not favor slow cooling from the annealing temperature if bright material is desired. Stainless iron undergoes selective oxidation in a furnace of the same type, and this oxide is not easily reduced. The paper is concluded by describing two different furnace designs for the continuous annealing of these materials. One is used for low-carbon steel, and the other for stainless iron. The furnace designs are based on the data obtained from the experiments described in the paper.

Inclusions and Their Effect on Impact Strength of Steel, I. By A. B. KINZEL and WALTER CRAFTS. (*Tech. Pub. No.* 402; also *Trans., Iron and Steel Div.*, 1931. 11,000 *words.*)—A study of normalized chromium-vanadium and nickel steel shows that for a given hardness the tensile-impact strength decreases as the inclusions increase. This loss of strength increases with greater hardness. A quantitative correlation of tensile-impact strength with total length of inclusions per unit area, counted at 50 magnifications, has been effected. This, together with the chemical analysis of inclusions, has emphasized the importance of nonmetallic matter, but also shows that while visible inclusions decrease the impact strength, body is so important that the effect of visible inclusions may be masked. A striking relation between visible inclusions, extracted inclusions, and sulfides has been found, the sulfur seeming to cause emulsification. The present commercial rating is shown to be unreliable, although satisfactory in many instances, and the possibilities of modification of specifications for cleanliness and quality are discussed.

Inclusions and Tensile-impact Strength of Steel, II. BY A. B. KINZEL and WALTER CRAFTS. (*Tech. Pub. No.* 436; also *Trans.*, Iron Steel Div., 1931. 5000 *words.*)—The use of the tensile-impact test for studying the dynamic quality of steel is described in this paper, the work being based on the principles set forth in a previous paper of the same title. Steels of various types, from various sources, and made in various ways, have been submitted to tensile-impact test and inclusion count. The correlated results show the great importance of the little known factors comprised in the term "body." These results also confirm the detrimental effect of inclusions previously reported. The tensile-impact strength is also a function of hardness and this relation has been worked out over a convenient range. From this work it is possible to set a practical lower limit of transverse impact strength as a measure of the dynamic quality of the steel for specification purposes and such a lower limit has been set as an illustration.

Some Notes on Blue Brittleness. By LELAND RUSSELL VAN WERT. (*Tech. Pub. No.*404; also *Trans., Iron and Steel Div.*, 1931. 5600 *words.*)—This is an inquiry into the causes of the peculiar results observed when low-carbon steels are subjected to torsion within the temperature range of 200°–300° C. The various possibilities are systematically considered, and the author comes to the conclusion that yielding and apparent elastic recovery are the most probable causes.

The Carbon-oxygen Equilibrium in Liquid Iron. By H. C. VACHER and E. H. HAMILTON. (*Tech. Pub. No.* 409; also *Trans., Iron and Steel Div.,* 1931. 6500 *words.*)—Two methods were used in attaining equilibrium within liquid iron at a given temperature and one atmosphere pressure. In the first method carbon was added to liquid iron containing oxygen. After equilibrium had been established the iron was solidified and analyzed for carbon and oxygen. In the second method liquid iron, of

various compositions, which had been previously outgassed in a vacuum, was exposed to carbon-oxides of known composition, solidified and analyzed. To prevent loss of carbon and oxygen on freezing some melts were killed with silicon and some with aluminum. Variations in results obtained by the first method made individual results doubtful. The products of the per cent carbon and oxygen obtained from ingots prepared by the second method at 1620° C. were constant. The composition of these ingots varied from 0.94 to 0.014 per cent carbon and from 0.003 to 0.196 per cent oxygen. The average of eleven ingots was 0.0025, with a mean deviation from the average for the individual ingots of ± 0.0003.

The Austenite-pearlite Transformation and the Transition Constituents. By ALBERT SAUVEUR. (*Tech. Pub. No.* 412; also *Trans., Iron and Steel Div.*, 1931. 2900 *words.*)—It is shown that the conversion of austenite into pearlite involves the allotropic transformation of the solvent, gamma iron, into alpha iron and the transformation of the dissolved carbon or carbide into separate particles of the carbide. Both these transformations are necessarily progressive, hence mixtures of gamma iron, alpha iron, dissolved carbon and precipitated carbide must exist before the pearlitic stage is reached. These mixtures correspond, in composition at least, to the transition constituents. It is concluded that austenite cannot transform into pearlite without transition constituents being formed.

Age Hardening of Austenite. By F. R. HENSEL. (*Tech. Pub. No.* 419; also *Trans., Iron and Steel Div.*, 1931. 12,500 *words.*)—This paper gives the results of a series of experiments on the precipitation hardening of certain austenitic alloys to which titanium and molybdenum were added as age hardeners. The changes of structure and of a number of physical properties during aging have been investigated. The tensile and elastic properties of the titanium alloys could be increased very materially. Although this increase is connected with a considerable loss of ductility, the final value of the latter is high enough to make the material very interesting from an engineering standpoint.

The Art of Metallography. By FRANCIS F. LUCAS. (*Tech. Pub. No.* 421; also *Trans., Iron and Steel Div.*, 1931. 18,000 *words.*)—The fundamentals of metallographic research were discussed by Mr. Lucas particularly for the benefit of the younger scientists who intend to follow this branch of research. After describing the working of the microscope, and the testing, correcting, and cleaning of objectives, Mr. Lucas went on to describe the new Zeiss metallurgical equipment. By means of this equipment it will be possible to achieve crisp, brilliant images at twice the present limits of useful magnification. An interesting description was also given of the ultra-violet research which was brought to a high stage of perfection at the Bell Telephone Laboratories by Mr. Lucas. In closing his lecture, Mr. Lucas presented a new theory of the cause of fatigue failure in hardened steel bases on the results of some of his more recent metallographic research. In this theory the claim was advanced that failures of this nature might be due to the presence of tiny cracks only a few hundred atom-diameters across and which he recently discovered in such steels. Due to the location of these cracks with respect to the martensite crystals found in the steel, it is believed that the cracks are produced by a very slight shrinking in the volume of the steel as it passes from one crystal stage to another during the hardening or tempering process.

Low-carbon Steel. By H. B. PULSIFER. (*Tech. Pub. No.* 426; also *Trans., Iron and Steel Div.*, 1931. 14,000 *words.*)—Simple carbon steel containing 0.10 per cent carbon, after making into cap screws, is studied in six different conditions: (1) annealed; (2) cold formed; (3) containing hard nodules; (4) holding dispersed troostite; (5) containing martensite and ferrite; and (6) largely martensitic. Typical properties and microstructures are given. The literature is reviewed. Six commercial low-carbon cap screws that cover a wide range of structures are studied briefly. On the basis of the preceding the brittleness in surface hardened low-carbon steel is discussed.

Comparison is made with wrought iron and Bessemer screw stock. The presence of hard nodules is considered a chief cause of brittleness in the low-carbon steels. It is thought that holding the carbon content low and a single high temperature quench are the two most vital conditions for producing tough cores. Reheating at just above the lower critical temperature is a contest between getting the case hard and causing nodular brittleness. Many new data are presented and the photomicrographs, especially, help to explain facts well known but not understood.

Magnetic Properties Versus Allotropic Transformations of Iron Alloys. By T. D. Yensen and N. A. Ziegler. (*Tech. Pub. No.* 427; also *Trans., Iron and Steel Div.*, 1931. 2300 *words.*)—Wever's classification of the elements based on their effect on the allotropic transformation of iron is compared with the classification of the elements based on their effect on the magnetic properties of iron. It is found that there is good agreement. An attempt is made to attribute both relations to the distortion produced by the elements on the iron space lattice.

Composition Limits of the Alpha-gamma Loop in the Iron-tungsten System. By W. P. Sykes. (*Tech. Pub. No.* 428; also *Trans., Iron and Steel Div.*, 1931. 2500 *words.*)—Alloys of iron containing tungsten up to 6 per cent by weight were heated for several hours at temperatures between 900° and 1400° C. On quenching from within this range the resulting micro-structures indicated the existence of two solid-solution phases. One of these presumably represents the face-centered cubic solid solution of iron with a tungsten content of some 3 per cent. The second is the body-centered cubic solid solution containing about 6 per cent tungsten. From a series of compositions and heat treatments within this range, the boundaries of the two-phase field are established. The boundary on the tungsten-rich side corresponds quite closely with that previously determined by thermal analysis.

Dilatometric Study of Chromium-nickel-iron Alloys. By Vsevolod N. Krivobok and Maxwell Gensamer. (*Tech. Pub. No.* 434; also *Trans., Iron and Steel Div.*, 1931. 11,000 *words.*)—This is a dilatometric study of stainless steels containing about 18 per cent chromium and varying amounts of nickel and carbon. The effect of nickel is to lower the temperature of the gamma to alpha change on cooling, and to oppose raising this temperature by prolonged heating as can be accomplished in alloys containing little nickel. Carbon has a similar effect. With 0.05 per cent carbon, between 4 and 8 per cent nickel is necessary to retain some austenite on quenching, but with 0.25 per cent carbon, only 2 per cent nickel is sufficient. The process of hardening these steels, by quenching and then immersing in liquid air or a mixture of solid carbon dioxide and acetone, has been studied dilatometrically. There is evidence for metastable equilibrium at these low temperatures between the chromium-nickel austenite and the hardening phase, carbides or alpha solid solution. The range of compositions n which hardening by this treatment may be accomplished is indicated.

Transformational Characteristics of Iron-manganese Alloys. By Howard Scott. (*Tech. Pub. No.* 435; also *Trans., Iron and Steel Div.*, 1931. 9500 *words.*) Investigation of low carbon Fe-Mn alloys by the dilatometric method showed that the transformations in the solid state are more complex than indicated by early work. A series of alloys covering the range between 5 and 24 per cent manganese in small steps was studied over the temperature range −180° to +800° C. The familiar transformations of iron, Ac_3 and Ar_3, occur at progressively lower temperatures with increasing manganese content until at about 13 per cent they disappear by suppression rather than depression of Ar_3 below atmospheric temperature differing in this regard from the Fe-Ni alloys. Coincident with the suppression of A_3, a new phase, epsilon iron (hexagonal closed packed crystal structure) identified by X-ray studies appears. This phase transforms into gamma iron on cooling with a contraction which distinguishes it from the transformation Ar_3 which is accompanied by an expansion.

The difference between the temperature of occurrence on heating and that on cooling is very much less than in the case of A_3 It disappears by suppression at about 23 per cent manganese. At higher manganese contents a reversible expansion anomaly similar to that in invar is found. An irreversible expansion occurs only on heating near 400° C. in alloys showing the A_3 transformation. This is attributed to the completion of the change at Ar_3 arrested on cooling by pressure effects. A contraction on heating only is identified with the precipitation of carbide from solid solution in an alloy of the Hadfield composition. Of special interest is the establishment of the locus of Ar_3 as a function of composition in Fe-Mn-Ni-C alloys because of its importance in gamma-iron metallurgy. The temperature at which Ar_3 starts is shown to be linearly related to the equivalent nickel content which is defined as the nickel content plus 2.5 times the manganese content plus 18 times the content of carbon in solid solution for Fe-Mn-Ni-C alloys over a wide range of compositions. Thus the temperature at which Ar_3 starts, T is given by:

$$T = 766 - 25.5 \text{ per cent Ni} + 2.5 \text{ (per cent Mn)} + 18 \text{ (per cent C)}$$

when T is less than 500° C. and the nickel content is over 5 per cent or the manganese content under 10 per cent.

The Development of Continuous Gas Carburizing. By R. J. COWAN. (*Tech. Pub. No.* 439; also *Trans., Iron and Steel Div.*, 1931. 9500 *words.*)—A hydrocarbon gas is mixed with flue gas containing CO_2 and introduced into a muffle containing work to be carburized so as to move through in the same direction as the work. In the preheat zone of the furnace the hydrocarbon gas begins to break down so that the work to be carburized soon becomes covered with a deposit of precipitated carbon. In the carburizing zone, active carburizing takes place due to the chemical reaction between this deposited carbon and carbon dioxide which at these temperatures becomes effective. In the subsequent diffusion zone the carbide case formed is allowed to penetrate toward the core to obtain proper gradation of properties. The metal in passing progressively through these three zones becomes carburized as desired by a proper regulation of the time, the gas flow and proportioning of the mixture, so that any carbon composition desired as a case may be consistently obtained. After completing this cycle, the metal may be quenched or cooled slowly as desired. This provides a new process for carburizing steel which is different from anything before attempted.

Method for the Electrolytic Extraction of MnO, MnS, FeS, and SiO_2 Inclusions from Plain Carbon Steels. By G. R. FITTERER. (*Tech. Pub. No.* 440; also *Trans., Iron and Steel Div.*, 1931. 7500 *words.*)—This paper describes in detail a method which if properly applied will satisfactorily determine the quantity of some of the more common types of oxide and sulfide impurities in plain carbon steels. The procedure may be divided into two main steps. The first involves the electrolytic solution of the steel sample in a chemically neutral ferrous sulfate-sodium chloride electrolyte. As the iron or steel is forced into solution, the non-metallic impurities fall to the bottom of a collodion bag surrounding the sample, where they are retained. After approximately 24 grams of steel are dissolved the inclusion residue is analyzed. The chemical analysis of the residue represents the second step in the procedure and is described in minute detail. Typical inclusion analyses of various steels are included. This paper was written primarily for the steel plant analyst who will find sufficient information therein to set up the apparatus and proceed with the electrolysis of the steel and the analysis of the residue.

Metallurgy

The World of Metallurgy. By JOHN A. MATHEWS. (*Preprint.* 6500 *words.*) This is an address delivered before a group of college seniors and first year engineering

students at Columbia University. Its purpose was to inform these young men as to what "metallurgy is all about" and also to interest them in the subject as a future profession. The article defines metallurgy and discusses briefly its two principal phases—first, extraction of metals from ore, and, second, fitting them for use. It cites some of the outstanding achievements of metallurgists in these two fields, such as the cyanide process for extracting gold, the work of Daniel C. Jackling, the "basic process" for extracting phosphorus in the production of steel. It also describes the latent possibilities which may be developed in steel and other alloys by heat treatment, the importance of research—both mining metallurgy and plant metallurgy—and the various ways in which research may lead to increased profits. In conclusion the writer discusses some of the phases of metallurgical education and emphasizes the necessity of acquaintance with the literature and with the broad principles underlying metallurgical training, such as are afforded by a thorough knowledge of chemistry, physics and mathematics, and a general acquaintance with the so-called "scientific" method of approaching and solving problems.

Institute of Metals Division

Plasticity of Copper-zinc Alloys at Elevated Temperatures. By ALAN MORRIS. (*Tech. Pub. No.* 390; also *Trans., Inst. Met. Div.*, 1931. 4000 *words.*)—A series of drop-hammer tests at elevated temperatures has been made on brasses ranging upward from 62 per cent copper. The alloys usually considered most difficult to hot work show the least plasticity. Alpha brasses which have been heated so as to develop a large grain appear to have a greater tendency to crack than samples of the same alloy which have not been so overheated. A method of calculating the average resistance of the sample to the blow is offered, which may prove to be a means of correlating the work of various investigators, though they have used different size samples and different weights of blow. The calculated resistances of lead, tin, aluminum and zinc at room temperature are compared with the plastic flow points, as determined by ordinary test. The relation between these two quantities varies with the different metals. The calculated resistance of lead, tin and zinc is much higher than the plastic flow points, while in the case of aluminum the difference is not so great.

Die Pressing of Brass and Copper Alloys. By JOHN R. FREEMAN, JR. (*Tech. Pub. No.* 391; also *Trans., Inst. Met. Div.*, 1931. 3900 *words.*)—The paper discusses the die pressing or hot forging of brass and copper alloys. The advantages of die pressings as compared to castings are discussed. The shearing of the "slugs" from extruded rods or shapes is described and the different types and relative advantages of crank, drop, and screw presses are discussed. The importance of die design is emphasized. Drawings are given illustrating typical types of dies for pressing of parts of various forms, and photographs are given of parts formed in the several dies described.

Studies upon the Widmanstätten Structure, II.—The β Copper-zinc and the β Copper-aluminum Alloys. By ROBERT F. MEHL and O. T. MARZKE. (*Tech. Pub. No.* 392; also *Trans., Inst. Met. Div.*, 1931. 15,000 *words.*)—The precipitation of the α and γ phases from the β in the Cu-Zn system and of the α from the β in the Cu-Al system have been studied. These alloys were of special interest since it was possible to precipitate phases of different crystal structure (α and γ) from the same basic lattice (β) and thus to decide unequivocally whether the structure of the parent lattice is solely determinative in the type of Widmanstätten structure obtained. It is shown that the α precipitate in Cu-Zn alloys is in the form of true needles nearly parallel to the [111] direction in the β lattice. The true direction of the needles is in the neighborhood of [556], thus exhibiting twelve families. The etching effects of the α needles show that the α lattice in each needle family bears the same relationship

in orientation to the β lattice. The γ precipitate on cooling or on quenching and reheating takes the form of stars, which on longer heating shrink into well-defined polyhedrons. The atom plane in the β lattice bounding these polyhedrons is the [110] plane, and the atom plane in the γ lattice is very probably also the [110] plane. Reasons are advanced for the formation of needles in the α precipitation and of dodecahedrons in the γ precipitation. The formation of these two distinctly different types of Widmanstätten figure adds further support to the previously enunciated theory (first paper of this series) that the type of Widmanstätten figure is not controlled entirely by the structure of the present solid solution, but by a cooperation between the lattices of the parent solid solution and of the precipitate. The precipitation of the α phase from the β in the Cu-Al system is closely analogous to that in the Cu-Zn system. Because of the far-reaching structural analogies in metal systems it is suggested that Widmanstätten structure analogies exist also.

Constitution of High-purity Aluminum-titanium Alloys. By W. L. Fink, Kent R. Van Horn, P. M. Budge. (*Tech. Pub. No.* 393; also *Trans., Inst. Met. Div.*, 1931. 8500 *words.*)—A survey of the existing literature showed that the constitution of the binary aluminum-titanium system had not been definitely established. The present investigation of this system included thermal, microscopic, and X-ray studies of a number of high-purity alloys. Cooling curves are not reliable for the determination of the liquidus because the thermal point representing primary crystallization in the region investigated is weak and lowered by under-cooling. Accurate results were obtained by determining the solubility of $TiAl_3$ in aluminum by chemical analysis of the supernatant melt. The constitutional diagram for aluminum-rich aluminum-titanium alloys is given. The aluminum-titanium constituent occurring as plates in the aluminum-rich alloys was chemically separated and found to have the composition represented by the formula $TiAl_3$. X-ray diffraction data were obtained by the powder, Laue, and rotating crystal methods. Diffraction patterns showed that $TiAl_3$ crystals are tetragonal with lattice dimensions of $a_0 = 5.424$, $c_0 = 8.574$, $\frac{c_0}{a_0} = 1.58$. The application of the theory of space groups to the X-ray data resulted in a unique solution of the atomic arrangement of $TiAl_3$, 4d-8 (a, b, c, d). This solution was verified by comparing the calculated and observed intensities of the powder reflections.

Effect of Combinations of Strain and Heat Treatment on Properties of Some Age-hardening Copper Alloys. By W. C. Ellis and Earle E. Schumacher. (*Tech. Pub. No.* 395; also *Trans., Inst. Met. Div.*, 1931. 6000 *words.*)—For the purpose of developing combinations of higher strength and conductivity than are obtainable by heat treatment alone in the age-hardening copper alloys, an investigation has been made of the effect of heat treatment and strain hardening on the properties in question. Hard drawing after heat treatment increased the tensile strength of a copper-nickel-silicon alloy containing 4 per cent of nickel plus silicon from 113,000 to 148,000 lb. per sq. in. The properties of the alloy were further improved after hard drawing by low-temperature aging. In this connection aging for 4 hr. at 300° C. increased the conductivity 19 per cent, without appreciably decreasing the tensile strength. Strain was found to accelerate precipitation in this system which is in line with observations on other dispersion hardening systems. Alloys of copper containing cobalt and silicon did not show as pronounced aging effects after hard drawing as did the copper-nickel-silicon alloys. As a result of the study a material suitable for use as an electrical conductor combining high strength with reasonably high conductivity has been developed. Representative values for strength of this material are 145,000 lb. per sq. in. combined with an electrical conductivity of 38 per cent of annealed copper.

A Thermodynamic Study of the Equilibria of the Systems Antimony-bismuth and Antimony-lead. By Yap, Chu-Phay. (*Tech. Pub. No.* 397; also *Trans., Inst. Met. Div.*, 1931. 9000 *words.*)—*Part I*, The Sb-Bi system for a long time appeared to

contradict the phase rule in that it has an anomalous solidus, always obtained by cooling down from the melt. Ôtani, by means of the resistance method, showed it to be a normal solidus characteristic of completely isomorphous systems. The writer has applied the thermodynamic laws of the depression of freezing point to the system in order to ascertain whether the system is amenable to this kind of treatment and to determine from the internal evidence the cause of the anomalous solidus. It is concluded from a study of the evidence that the cause of the *non-variant* crystallization is probably due to the formation of Bi_3, although antimony and bismuth should normally be considered diatomic. The heats of fusion of antimony and bismuth are calculated to be 20.0 and 14.2 cal. per gram, which is in agreement with the most reliable experimental values obtained by Umino. *Part II.* The lead-rich end of the Sb-Pb system has been subjected to a similar thermodynamic analysis. In this range, antimony is dissolved in molten lead in the monatomic form, although it is normally dissolved in solid solution in lead as Sb_2 down to about 150° C. Below that temperature, antimony is dissolved in the monatomic form. This suggests a transformation either of antimony or the lead solid solution at around 150° C. (probably less). The heat of fusion of lead is calculated to be 5.7 cal. per gram, which is likewise in good agreement with reliable experimental values. Attention is called to a new method of calculating the heat of fusion of a solute.

Forming Properties of Thin Sheets of Some Nonferrous Metals. By W. A. Straw, M. D. Helfrick and C. R. Fischrupp. *Tech. Pub. No.* 406; also *Trans., Inst. Met. Div.,* 1931. 6800 *words.*)—A simple method of determining the "formability" of sheet brass, phosphor bronze and nickel silver which duplicates actual manufacturing conditions is described. Data of direct application in selecting grades, tempers and thicknesses of material for design and manufacture are given in terms of minimum radii of 90° forming tools which produce bends without cracking the surface of the materials. Examples of the application of the results in connection with the design of formed parts and forming tools are shown.

Directional Properties of Cold-rolled and Annealed Copper. By Arthur Phillips and E. S. Bunn. (*Tech. Pub. No.* 413; also *Trans., Inst. Met. Div.,* 1931. 6000 *words.*)—This paper records the directional differences in the tensile properties of two kinds of sheet copper, namely, tough pitch copper containing 0.0295 per cent silver and electrolytic copper deoxidized with phosphorus (residual phosphorus 0.0089 per cent). The first series of tests was on the cold-rolled copper, reduced 10 to 90 per cent. Tensile tests were made on strip cut (1) in the direction of rolling, (2) at 90° and (3) at 45° to the direction of rolling. In the second series of experiments, the three kinds of strips cut from the rolled sheets were annealed over a temperature range extending from 300° to 800° C. The results seem to indicate that rather pronounced directional differences, particularly in elongation values, are produced by the combined effects of heavy reductions and high temperatures of anneal.

Influence of Stress on Corrosion. By D. J. McAdam, Jr. (*Tech. Pub. No.* 417. 19,000 *words.*)—In this paper, special attention is given to nickel, aluminum-bronze, stainless iron, nitrided steel and Muntz metal. Each experiment was in two stages: (1) a corrosion stage, in which the specimen was corroded with or without cyclic stress; (2) a fatigue stage, in which the specimen was tested to fatigue failure and the fatigue limit was estimated. The lowering of the fatigue limit is used as a measure of the "damage" due to corrosion. Diagrams of three types are presented: Type 5 illustrating relative influence of stressless corrosion and corrosion under cyclic stresses, in causing damage; Type 10 illustrating influence of cycle frequency on net damage; and Type 11-a illustrating influence of stress range. These diagrams, in addition to diagrams for steels, aluminum alloys and monel metal presented in previous papers, illustrate the behavior of a great variety of metals under similar conditions of corrosion. The rate of net damage varies as the third to at least the fifth power of the

corrosion stress, depending on the metal, cycle frequency, and corrosion conditions. The general conditions favoring intercrystalline corrosion are discussed. Application of stress-corrosion data to design, construction and operation of machinery and structures is discussed briefly.

Some Important Factors Controlling the Crystal Macrostructure of Copper Wire Bars. By L. H. DeWald. (*Tech. Pub. No.* 429. 5200 *words.*)—Extant literature contains little reference to the methods of production and commercial preferences of the various crystal structures found in copper wire bars. An investigation was therefore conducted at the Hawthorne Works of the Western Electric Co. to determine the casting factors responsible for variation in the crystal structure of copper wire bars. Beginning with copper of a uniform chemical composition and closely controlled gas content, experimental wire bars of the same bar to mould weight ratio as the larger commercial bars were cast in an experimental mould in which all the casting factors could be varied at will. Conclusions show that the casting factors to be controlled to produce a definite crystal macrostructure are, in order of importance: (1) molten metal temperature, (2) casting speed, (3) mould temperature. Photomacrographs show the extent to which these factors influence the resulting crystal structure. A significant conclusion arrived at is that copper produced at low temperatures has been found, in general, to be preferable for producing copper wire-bars to be converted into fine wire, and since a fine crystal macrostructure of the molten copper is indicative of the use of low temperature, those wire-bars having this fine crystal macrostructure are considered preferable for wire drawing purposes.

Some Developments in High-temperature Alloys in the Nickel-cobalt-iron System. By C. R. Austin and G. P. Halliwell. (*Tech. Pub. No.* 430. 10,000 *words.*)—The paper deals with the mechanical properties at elevated temperatures of nickel-cobalt-iron base alloys containing a few per cent titanium. A limited number of alloys also contain additions of a fifth element. The properties of these alloys were examined by use of the conventional high temperature tensile test, by a new high temperature bend test, and by Vickers hardness tests on the alloys aged at various temperatures for a maximum of 3000 hours. It is suggested that marked aging characteristics of the alloys is due to the presence of titanium. Data have been provided illustrating that in many of the alloys about 80 per cent of maximum age hardening is completed after 72 hr. at 650° C. Prolonged aging at this temperature does not usually lead to a decrease in the hardness of the fully aged alloy. The tensile tests reveal unusually high values for the ultimate strength and proportional limit at 600° C.

Seasonal Variation in Rate of Impingement Corrosion. By A. Morris. (*Tech. Pub. No.* 431. 4500 *words.*)—Impingement corrosion tests were undertaken in the spring of 1929 and have been carried on up to the present time. The water used was pumped directly from an estuary and through the apparatus. Pairs of alloys were compared by subjecting ten samples of each to impingement corrosion, simultaneously and under as nearly similar conditions as possible. The average depth of attack formed the basis of comparison. During the summer a marked increase in the rate of attack has been experienced. The magnitude of this increase is greater than the observed difference between any two alloys compared. Results of individual tests are given.

Age-hardening Copper-titanium Alloys. By F. R. Hensel and E. I. Larsen. (*Tech. Pub. No.* 432. 3000 *words.*)—This paper describes some age-hardening copper-titanium alloys. Hardness, electrical conductivity and tensile properties were determined. A tentative constitutional diagram of copper and titanium up to 30 per cent Ti has been constructed from thermal and X-ray data.

The Equilibrium Diagram of the Copper-rich Copper-silver Alloys. By Cyril Stanley Smith and W. E. Lindlief. (*Tech. Pub. No.* 433. 8000 *words.*)—The equilibrium diagram of the copper-silver system from 0 to 12 per cent silver has been

redetermined. The liquidus agrees well with previous investigations, while the solid solubility, determined by the microscopic examination of quenched samples, was found to be 7.9 per cent at the eutectic temperature, 779.4° C. The solubility decreased rapidly below this point, becoming 4.4 per cent at 700°, 2.1 per cent at 600°, 0.90 per cent at 500° and about 0.4 per cent at 400° C. and below. This solubility curve is in good agreement with those recently published by Stockdale and by Ageew, Hansen and Sachs. The age hardening which would be expected to result from this change in solubility was not realized, and the tensile strength actually decreased during precipitation, although high electrical conductivity was obtained in alloys annealed at low temperatures.

Preparation of Graded Abrasive for Metallographic Polishing. By J. L. Rodda. (*Tech. Pub. No.* 438. 4500 *words.*)—Commercial abrasives have been found unsatisfactory for polishing zinc due to the wide range in size of their particles. A method of separating such abrasives into portions having a limited uniform range of sizes was worked out. This work was carried out mainly with alumina and emery and the size of the particles was checked by microscopic examination. The essential steps in the process are as follows: (1) Thorough dispersion of the abrasive in water. This is accomplished by using a small amount of sodium silicate as a peptizing agent and dispersing in either a colloid mill or pebble mill. (2) The abrasive suspension is allowed to settle for a definite time and siphoned off to a predetermined depth, using the siphoned material. Photomicrographs illustrate the abrasives obtained by this process. Photomicrographs of plated materials and zinc alloys polished with these abrasives are also shown.

Copper Embrittlement. By L. L. Wyman. (*Preprint.* 5300 *words.*)—This paper deals with a condition wherein copper is alternately subjected to an oxidizing action, between 400° and 900° C., and a reducing action, between 500° and 800° C., in a process which cannot be changed to suit the material, but for which the material must be selected to withstand the process. Several different types of commercially procurable coppers, including commercial copper, vacuum copper, and coppers deoxidized with silicon, calcium boride, zinc, and silicon and phosphorus, are subjected to a series of treatments encompassing all the manufacturing conditions. These materials have been classified according to the penetration of the cracking, or depth of embrittlement. The zinc and silicon deoxidized coppers are the most resistant to cracking, followed by the calcium boride, vacuum, silicon and phosphorus, and commercial coppers, respectively. The results show that the resistance is dependent on the purity of the copper and the kind of deoxidant used, but that supposedly duplicate lots vary considerably.

Relation of Crystal Orientation to the Bending Qualities of a Rolled Zinc Alloy. By G. Edmunds and M. L. Fuller. (*Preprint.* 5200 *words.*)—The relation of crystal orientation to the bending properties of pure and alloyed zinc is discussed in the light of the explanation generally accepted for the mechanism of the plastic deformation of zinc. The two types of preferred orientation unfavorable for bending are deduced. Experimentally one of these types is observed as a thin layer at the surface of strips of a rolled zinc alloy. Poor bending qualities are found whenever this orientation persists to a depth of 0.0005 in. or more. Confirmation of this relationship is obtained by determining the effect on bending properties of the removal by etching of the surface layer. From a study of orientation beneath the surface, the cause of the better across-grain than with-grain bending properties is determined.

The Beta to Alpha Transformation in Hot-forged Brass. By Robert S. Baker. (*Preprint.* 2000 *words.*)—This paper describes the beta to alpha transformation as it occurs in hot-forged brass, and points to certain conditions under which the conversion may occur during hot pressing. In a brass forging, normally having an alpha-beta structure, the all-alpha fields were found along the lower or die side of the sample.

After considering the factors involved, and concluding that a steel die and hammer may serve as a quenching medium, the occurrence of these all-alpha fields was attributed to the beta to alpha transformation. The transformation was found in all cases in a forging containing approximately 60.30 per cent copper and 1.75 per cent lead. The rod was heated above 800° C. before forging and the die, which acts as a quenching medium, did not have to be cold.

Metal Working in Power Presses. By E. V. CRANE. (*Preprint; also Trans., Inst. Met. Div.*, 1931. 15,500 *words.*)—Following the division of common power-press operations into the four very general groups of shearing, bending, drawing and squeezing, the paper proceeds to describe the principal stresses and the metal movements distinguishing the groups and three or four outstanding subgroups. The changes occurring are illustrated by sketches, samples, test curves and photomicrographs. All of the experiments were made in nonferrous metals, paralleling ferrous tests in results. "Rate of strain hardening" is suggested as a basis for gaging previous cold working and predicting results of operations to be performed. Tentative curves are offered for copper and Tobin bronze. The latter was prepared from a combination of tensile and compressive tests of annealed and unannealed material. These were plotted against *actual* unit stress on one coordinate and percentage reduction, as the measure of distortion, on the other coordinate. An interesting comparison of structure is shown of two cartridge cases, one produced by drawing and ironing, the other by extrusion. With particular respect to strain hardening, the recrystallization range is taken as the proper division between hot and cold working. Kent's instructive forgeability curves are reproduced in a brief outline of hot press forging, especially of beta brass.

X-ray Determination of Alloy Equilibrium Diagrams. By ARNE WESTGREN. (*Preprint;* also *Trans., Inst. Met. Div.*, 1931. 12,500 *words.*)—The lecture deals mainly with the identification of alloy phases and the determination of their homogeneity ranges by means of X-ray investigation. It also discusses the question of the difference between intermetallic compounds and solid solutions. A short survey of the structure analogies of alloy phases is given.

Suppressed Constitutional Changes in Alloys. By G. SACHS. (*Preprint;* also *Trans., Inst. Met. Div.*, 1931. 4000 *words.*)—This paper deals with changes in solubility and polymorphic transformations with particular reference to phenomena of the age-hardening type. A thermodynamic viewpoint is developed for examining such transformations.

Texture of Metals after Cold Deformation. By FRANZ WEVER. (*Preprint;* also *Trans., Inst. Met. Div.*, 1931. 12,000 *words.*)—Starting with the conception that only a complete and unprejudiced interpretation of the X-ray data can appropriately serve as the basis for the deduction of the relationships between the orientation of the crystallites in a cold-worked metal and the inner slip mechanism of that metal, a new method for describing conditions of statistical anisotropy with the aid of pole figures is described. The simple relationships of these figures to the X-ray diffraction patterns are shown and an example is discussed which shows the conversion of the X-ray patterns to pole figures by means of graphic charts. The formation of the texture by drawing, cylindrical compression, rolling and plane parallel compression is described by means of pole figures for the limited case of cubic metals with face-centered and body-centered lattices. The known laws of plastic deformation of face-centered cubic single crystals are used to elucidate the deformation process of a crystallite in a structure. In this, particular importance was given to the mechanism of the plastic bending of the crystal. Systematically following the changes in position of randomly orientated crystallites in an axially symmetrical deformation process leads to a texture which is found experimentally to contain those crystallite orientations, in stable or labile positions, for which no further lattice rotation occurs. The

texture of aluminum after plane parallel deformation is described as a superposition of parts of textures produced by axially symmetrical tension and compression with their axes respectively in the direction of flow and of compression of the deformation process. These two propositions are extended to the body-centered cubic structure under the assumptions that the (011) plane is the plane of slip and that the direction 111 is the direction of slip.

Crystallite positions found by experiment to differ from those to be expected from theoretical considerations were investigated by means of various deformation textures of aluminum and the findings were accounted for by means of the processes of deformation and strengthening of single crystals.

Inasmuch as plane parallel deformation is closely related to the rolling process, which is also confirmed by the similarity of their textures, a unified viewpoint is reached from which to consider the important deformation textures. Its basis lies in the well-known mechanism of deformation of single crystals.

Fabrication of the Platinum Metals. By C. S. Sivil. (*Preprint;* also *Trans., Inst. Met. Div.,* 1931. 6500 *words.*)—The early method of metallizing platinum sponge is reviewed, also the melting by oxyhydrogen blowpipe, arc melting, induction melting and melting by atomic hydrogen flame. The effect of oxygen and hydrogen on platinum, the effect of various refractories, especially lime, and ingot molds are described and discussed. Methods and difficulties of working the metals are reviewed, and certain special applications are mentioned. Platinum and palladium are said to be easily worked, rhodium with difficulty, iridium with greater difficulty, and ruthenium and osmium not at all.

Coal Division

Subsidence and Ground Movement in a Limestone Mine and on the Surface, Caused by Longwall Mining in a Coal Bed Below. By R. Laird Auchmuty. (*Tech. Pub. No.* 396; also *Trans., Coal Div.,* 1931. 9500 *words.*)—This paper presents some of the engineering data collected and prepared for the Marquette Cement Manufacturing Co., at La Salle, Ill., and used as evidence in a suit with the Oglesby Coal Co., in which the cement company prevented further undermining of its property by the coal company. The cement company was mining by room and pillar methods a 30-ft. limestone bed 125 ft. under the surface and the coal company was mining by longwall methods a 42-in. coal bed, underlying the limestone 435 to 470 ft. On three plates are plotted for various time intervals the location of points along three base lines, with accompanying survey records, showing the vertical, longitudinal, and lateral movement. Two lines are on the surface and one in the limestone mine. On another plate are recorded the angles of draw observed along the several base lines. A study of the records shows that in nearly all cases subsidence commences far in advance of the longwall face, and that preceding subsidence there is a slight upthrust. Observation of longitudinal and lateral movement shows that this action is always oscillating in character. Survey records show that subsidence is very irregular in relation to time and that slight upheaval may occur after subsidence has started.

Effect on Buildings of Ground Movement and Subsidence Caused by Longwall Mining. By Wallace Thorneycroft. (*Tech. Pub, No.* 398; also *Trans., Coal Div.,* 1931. 9500 *words.*)—This paper deals with two cases: The first relates to the effect on a massive century-old stone residence at Plean, Scotland. A 5-ft. bed had previously been taken out by advancing longwall with pack walls. This study dealt with the mining of a coal bed 20 in. thick, 90 ft. above the worked-out 5-ft. bed and about 500 ft. below the surface. Precise data were obtained of the subsidence with relation to time and the advance of the longwall face. Cracks opened and again closed in the stone residence. Long plumb lines were suspended from the eaves at the corners of the house. The angle of the plumb lines gave a measure of the slope of the subsidence

wave as it approached, passed under, and went beyond the residence. The second case dealt with the effect of mining a 2-ft. bed at a depth of 408 ft. on a massive water tower. Plumb lines were suspended from supports at the top of the tower. In this case apparently no damage was done to the tower, although the solid rock on the surface was "whinstone" (eruptive rock). Although advancing longwall is little used in the United States, the effect is not dissimilar from retreating longwall which is being more and more used. The difference, however, is that in retreating longwall the goave is seldom systematically packed. Hence the effects as regards subsidence are undoubtedly greater than where tight packing is done.

Subsidence in the Sewickley Bed of Bituminous Coal Caused by Removing the Pittsburgh Bed in Monongalia County, West Virginia. By S. D. Brady, Jr. (*Trans., Coal Div.*, 1931. 2000 *words.*)—Several cases of simultaneous mining in the Sewickley and Pittsburgh beds are described, and the conclusions are drawn that: (1) the Sewickley and Pittsburgh beds of coal can be mined simultaneously in an economical and successful manner, with average recovery; (2) the Sewickley bed can be removed before the Pittsburgh bed with no harm to the mining of the latter, if the Sewickley bed is completely and properly removed before the pillaring in the Pittsburgh bed is undertaken; (3) the Sewickley bed can be worked successfully, where the intervening strata are of a soft nature and the Pittsburgh bed has been completely removed, with the additional cost of extra timber.

Instantaneous Outbursts of Carbon Dioxide in Coal Mines in Lower Silesia, Germany. By P. A. C. Wilson. (*Preprint;* also *Trans., Coal Div.*, 1931. 6500 *words.*)—The geology of the Lower Silesian coal basin is described and the origin, retention and release of the carbon dioxide are considered. Precautionary methods—exploratory drill holes, shock blasting, safety doors, ventilation—are discussed and a detailed description is given of the Wenceslaus mine and the instantaneous outburst of July, 1930.

Introductory Notes on Origin of Instantaneous Outbursts of Gas in Certain Coal Mines of Europe and Western Canada. By George S. Rice. (*Preprint;* also *Trans., Coal Div.*, 1931. 6000 *words.*)—Various occurrences of instantaneous outbursts are reviewed, conjectures are made regarding the origin of the hydrocarbon gases, Graham's and Briggs' tests of gas solubility in coal and Greenwald's diffusion tests are discussed, also the tests reported in 1927 by the Prussian Mining Department. The writer's conclusions regarding causes and his recommendations regarding precautions are given in detail.

General Review of United States Bureau of Mines Stream-pollution Investigation. (*Trans., Coal Div.*, 1931. 4500 *words.*)—This paper is a presentation of some general facts and information gathered during the past five years on coal-mine drainage. A stream in a district thought to be representative of those in low to average sulfur bituminous coal fields and another in a high-sulfur district were selected for study of variations of mine-waste waters as to acidity in the two districts, the possibility of sealing abandoned or worked-out mines or parts of mines and influence of outside "gob" piles.

Air Cooling to Prevent Falls of Roof Rock. By J. H. Fletcher and S. M. Cassidy (*Tech. Pub. No.* 387; also *Trans., Coal Div.*, 1931. 6000 *words.*)—Air has been heated, cooled, washed, humidified, and dehumidified at many metal and coal mines for various purposes, such as reducing the unbearable heat and high humidity of deep workings, allaying dangerous siliceous dust, and decreasing the hazard of a dust explosion in coal mines by adding moisture to the intake air. But at the operation of the Saxton Coal Mining Co., near Terre Haute, Ind., air is cooled for a rather unusual purpose—to prevent roof falls. At this mine, like many others, the roof is ordinarily very good except during each warm season when it deteriorates and comes down. At Saxton these frequent falls seriously interfered with haulage, trolley wires were torn down,

airways became choked with rock, and danger to the workmen was considerably increased. Therefore a plant was installed to cool all intake air during the warm months to a constant temperature low enough not to affect the roof. The cooling plant has been in operation for over three years, and favorable action of cool air on the roof has fully met all expectations. This is best described by stating that the roof is now equally as good in summer as in winter. Cost of operating the plant has been found to be relatively low, and even this cost is more than offset by the resulting lower deadwork cost, besides other benefits, such as increased safety.

The Roof of the Pittsburgh Coal Bed in Northern West Virginia. By LEE M. MORRIS. (*Trans., Coal Div.*, 1931. 2500 *words.*)—Obviously there were several changes during the period of deposition of the material overlying the Pittsburgh bed. Locally, sandstone occurs above and in contact with the coal, sometimes a shale or clayey shale rests on the bed and is overlain by sandstone. Throughout the greater portion of northern West Virginia, the basal member of the immediate roof is a black or gray clay, usually slickensided. The variations as they occur in different counties, and the problem of roof support are discussed. The writer concludes that systematic timbering is necessary to prevent accidents in mines with such dangerous conditions.

Mechanical Mining. By EUGENE MCAULIFFE. (*Trans., Coal Div.*, 1931. 3000 *words.*)—The writer prefers the term "mechanical loading" as more nearly correct in reference to present practice. He gives comparative figures for mechanical and hand mining in England and in the United States and discusses American experience and the need for study and consideration of improved systems of organization for the use of more machinery and the further extension of human engineering.

Comparison of Accident Hazards in Hand and Mechanical Loading of Coal. By EUGENE MCAULIFFE. (*Tech. Pub. No.* 399; also *Trans., Coal Div.*, 1931. 2900 *words.*)—The effect of increased use of machinery in coal mining on the accident hazard is an important matter and careful data such as this paper gives are of great value. Compensatable accidents are taken as the basis of the study and the results of 116,685 man-shifts worked with hand loading and 129,115 man-shifts with mechanical loading are given in comparative tables and show that mechanical loading gave about a 50 per cent increase in the man-shifts worked per fatal accident, while the tonnage loaded shows practically a 100 per cent increase. Fatal accidents, shown separately, also show an improvement as well, though the numbers are too small to be of more than suggestive value. The author concludes that under proper supervision mechanical loading of coal may afford an increased measure of safety to the workers as well as better operating results.

Selection of Mechanical Car-loading Equipment. By C. C. HAGENBUCH. (*Trans., Coal Div.*, 1931. 2000 *words.*)—In general, it is profitable to install mechanical coal-loading equipment: (1) when existing hand-loading contract rates prevent further cost reduction; (2) when the height of the coal bed requires that either top or bottom must be taken to place cars in rooms; (3) when labor shortage or limited house-plant capacity prevents the production of desired tonnage; (4) when tonnage requirements are in excess of possible hand-loaded production from developed areas; (5) when working places are scattered and it is possible to concentrate production, and therefore supervision, by the installation of mechanical equipment; (6) when rapid development is essential. The choice lies between mobile track equipment, pit-car loaders, scrapers, shaker conveyors, belt conveyors and drag conveyors. The major factors influencing choice of equipment are grades, thickness of bed, nature of pavement, nature of roof, mining systems, gassy or nongassy mines, fine dust stirred into suspension, impurity bands, size of product, impurity extraction at face, structure of coal, size of mine car, cutting machines available, maintenance cost, effect of breakdowns on output, rate of advancement, tonnage increase per man, loading rates, organization, cost credits and debits.

Stripping in the Anthracite Region. By H. H. Otto. (*Trans., Coal Div.*, 1931. 4000 *words.*)—A brief review of the status of anthracite stripping methods from 1917 to the present, and a detailed description of the stripping operation of The Hudson Coal Co. at Clinton colliery.

Mining Coal by the Stripping Method, with Particular Reference to the Operations of the Enos Coal Mining Co., Oakland City, Indiana. By Fred S. McConnell. (*Preprint.* 2200 *words.*)—A résumé of practice in contouring, drilling, blasting with liquid oxygen, stripping, loading, hauling and preparation, and tree planting to cover the spoil banks.

Premature and Hangfire Explosions in Anthracite Mines. By Charles W. Wagner. (*Trans., Coal Div.*, 1931. 2000 *words.*)—Causes of premature explosions lie in the methods of firing, namely: (1) cap and fuse with dynamite, (2) fuse with pellet powder, (3) squib with black powder or pellets, (4) exploders and delays with dynamite or black powder, (5) exploders with dynamite, (6) delay ignitors with black powder or pellets, (7) electric squibs with black powder or pellets. The three principal causes of hangfire explosions are (1) improper cleaning of the drill hole, so that drillings and fine dust collect between the cartridges of explosive when charging; (2) weak detonators; (3) insensitive explosives. Practically all of the accidents caused by firing could be avoided if the miner followed safe practices. Much more can be accomplished by an intensive educational program under the direct supervision of the foreman or section foreman in charge of the particular mine or miners than by laws. Discipline should only be resorted to when a miner persists in violating the instructions of his boss, or where discipline will have a good effect upon all of the men at the colliery. The mere making of rules and the application of discipline without education will make it necessary to police every workman in the mines.

Economic Aspects of Bituminous Coal Losses in Ohio, Pennsylvania and West Virginia. By James D. Sisler. (*Trans., Coal Div.*, 1931. 3000 *words.*)—The author discusses the six general causes of loss of coal in mining: (1) coal left on the roof and bottom; (2) coal lost in room, entry and panel pillars; (3) coal lost in oil-well or gas-well pillars; (4) coal lost under buildings, railroads and boundaries; (5) coal lost in handling and preparation, underground and surface; (6) coal lost by rolls, thin or dirty areas and under streams. By comparative figures he shows that the percentage of recovery in Ohio has not changed since 1922, in Pennsylvania it has increased 1.5 per cent and in West Virginia, 0.6 per cent. He predicts that the percentage of recovery will increase slightly in Ohio during the next 10 years and that the recovery in Pennsylvania and West Virginia will increase gradually.

Operating Organization at Mines of Consolidation Coal Co. By A. R. Matthews. (*Trans., Coal Div.*, 1931. 2000 *words.*)—A description of the function of the members of that portion of the organization of the Consolidation Coal Co. that is charged with the responsibility of the actual operation of an individual mine, showing its relation to the division personnel. Illustrated by a chart.

Measuring Mine Costs and Production. By N. A. Elmslie. (*Trans., Coal Div.*, 1931. 2000 *words.*)—A general discussion of the study of a mine with reference to costs and production, under the heads of ventilation, drainage, transportation, power and revision of mining methods.

Relation between Mine Performance and Mine Cars. By D. L. McElroy. (*Trans., Coal Div.*, 1931. 6000 *words.*)—Data taken from a study of mine haulage conducted by the School of Mines of West Virginia University at 42 mines in 8 coal fields of West Virginia. The paper deals with the number of mine cars in use per loader, the capacity of mine cars, and the distribution of mine cars.

Coal Evaluation and Preparation. By T. F. Downing, Jr. (*Tech. Pub. No.* 420. 3000 *words.*)—This paper stresses the need for more detailed study of coal beds and surrounding conditions in order that maximum efficiencies in uses of the fuel and

greatest possible realization may be obtained. The coal in a seam may change materially in structure and chemical content within a short distance so that any application of grade is necessarily local. The chapter on Mining Methods gives three instances wherein changing of mining practices to conform with data obtained from face tests changed the commercial possibilities of the product, gave chemical uniformity in shipments, and saved considerable outlay of capital expenditures for mechanical preparation plant and equipment. Under the heading Mechanical Preparation suggestions are given concerning advisable tests to decide the proper amount and the sizes to be treated and to aid in choice of equipment. When coal from two or more beds is treated in the same plant there is a possibility that proper segregation during preparation may bring better results and less loss of coal in the reject. The chapter on Ash Fusibility discusses the possibility of face tests showing the parts of the seam from which low fusion temperatures can be expected and those which may be rejected in mining. To sum up, a complete knowledge of the bed, or beds, is necessary before definite assignment to a grade can be applied to any particular coal. Correct application of mining methods and preparation is necessary before commercial shipments will substantiate the grade given.

Growth of Coal Preparation in the Smokeless Fields of West Virginia. By T. W. Guy. (*Tech. Pub. No.* 437. 4500 *words.*)—Coal was first shipped from the New River field in 1871, from the Pocahontas field in 1883, and the Winding Gulf field in 1907. Early preparation was confined to handpicking and screening by gravity. The slack was largely used for coke making. Some large lump was shipped as early as 1888, egg about 1890, and nut coal about 1900. The installation of shaking screens, picking tables, loading booms, etc., for the preparation of large sizes proceeded rapidly after the first installation at Landgraff, W. Va., in 1908. The first washer was installed at McComas, W. Va., in 1903 for slack coal. Up to the end of 1910, washers had been installed at 14 mines which produced in that year 14.7 per cent of the output from the Pocahontas field. Only one new plant was installed from 1910 to 1922. In 1923, the first plant cleaning coal by air was installed. Since 1923, the number of both wet and dry cleaning plants has increased rapidly, not only in the Pocahontas field, but the New River and Winding Gulf fields also. Early in the present year there were 73 cleaning plants belonging to 54 companies in the smokeless fields. These plants produced 42.5 per cent of the total smokeless output in 1930. It is striking to note that the first 14 mines which installed cleaning plants and which produced 14.7 per cent of the output in 1910 produced 14.4 per cent of the total output in 1920, 17.9 per cent in 1921, and 15.3 per cent in 1930. In 1921, the year of depression following the boom demand and prices of 1920, these mines increased their output 9.2 per cent, while the other mines in the Pocahontas field decreased their output 15.8 per cent. For the future, greater uniformity in the products shipped, coordination of preparation and marketing plans, and standards for preparation and marketing should receive careful consideration.

Conditioning of Coal for Treatment by Pneumatic Cleaners. By Thomas Fraser and Robert MacLachlan. (*Trans., Coal Div.,* 1931. 5000 *words.*)—The difficulties in the practice of dry cleaning of coal have been mainly in the auxiliary operations of conditioning the feed and handling the products. Major auxiliary problems are: (1) proper sizing and mixing of the feed; (2) handling wet coal and coal varying in condition; (3) dust handling; (4) provision of constant uniform feed.

Handling wet coal is the principal difficulty remaining to be solved. Experience with Pittsburgh coal indicates that coal of 5 per cent or less moisture content above this natural vein moisture may be screened satisfactorily down to as fine as ¼ in. Predrying of the raw coal to supply a uniformly conditioned feed would greatly improve the average performance of pneumatic tables. Tables operated with con-

tinuous even feed make a much better separation than those working intermittently on a fluctuating feed.

Combination Wet and Dry Coal-cleaning Process. By RAY W. ARMS. (*Trans., Coal Div.,* 1931. 3000 *words.*)—The purpose of this paper is to call attention to the broader significance of the term used in the title—that is, an application of the sized-coal cleaning principle adapting itself in an ideal fashion to present day multiple-size preparation practice—and to mention some of the factors that govern its application. The combination limits moisture in cleaned coal.

Operation of Rheolaveur Plant at Dorrance Colliery, Lehigh Valley Coal Co. By EDGAR SCHWEITZER. (*Trans., Coal Div.,* 1931. 4000 *words.*)—The reasons for installing a wet-washing plant at Dorrance colliery, with a detailed description of the plant and its operation.

Dust Collection in Pneumatic Cleaning Plants. By CHARLES H. J. PATTERSON. (*Trans., Coal Div.,* 1931. 2200 *words.*)—A résumé of experiments since the first noteworthy commercial installation of pneumatic equipment in 1922, including the Crane Creek plant of the American Coal Co., the Birtley process in England, the American Blower, Pangborn dust arrester and Peale-Davis system.

Economic Utilization of Natural Gas. By RALPH E. DAVIS, HARRY K. IHRIG, DEWVEY J. SABIN AND LYON F. TERRY. (*Trans., Coal Div.,* 1931. 10,000 *words.*)—This paper describes the mechanism of combustion of various types of fuels and indicates that the experimentally determined calorific value of a fuel is not the correct criterion for judging its value in comparison with other fuels and that the net available heating value, after subtracting heat not available for useful work, is the true measure of its worth. It shows that other factors, such as flexibility of control and costs of maintenance and repair, enter into any study of the economic utilization of natural gas. It gives equivalent prices of various coals and fuel oils as compared to Monroe natural gas, based on the net available heating values of the three fuels, and lists large-scale commercial tests of natural gas as compared with other fuels.

The Low-volatile Coal Field of Southern West Virginia. By H. N. EAVENSON. (*Tech. Pub. No.* 441. 12,000 *words.*)—The low-volatile coal field of southern West Virginia is in six counties of that State and two counties of Virginia. Characteristics of nine seams being worked are described, and analyses and reserves of these seams, over 3 ft. in thickness and having less than 23 per cent volatile matter, are given, with maps. Paper is entirely from an engineering, and not a geological, viewpoint.

Are Mine Accidents More Frequent in Low-volatile than in High-volatile Mines? (*Preprint.* 11,000 *words.*)—EXPERIENCE IN WEST VIRGINIA, by R. F. ROTH. The comparison developed by the author is based on 12 graphs covering the 10-year period from 1921 to 1930, plotted from data furnished by the West Virginia State Department of Mines. It covers fatal and nonfatal accidents from falls of roof and coal, mine cars and motors, and from all other causes. The general trend of fatal accident frequency in the mines of West Virginia during the 10-year period 1921 to 1930 has been favorably downward, as against the decidedly upward trend of nonfatal accident frequency; the general trend of frequency of total accidents, fatal and nonfatal, has been upward, because of the overshadowing influence of the nonfatal class. The general trend of fatal accidents of all kinds in the United States has been appallingly upward during the 10-year period. Ten of the graphs show a difference of accident frequency unfavorable to the low-volatile field. In conclusion the author asks whether a campaign of systematized effort, and research would not accomplish much for the coal industry of West Virginia.

EXPERIENCE IN PENNSYLVANIA by RUSH N. HOSLER. The mines of the Anthracite Region were excluded from this study because the methods of mining and the geological conditions are not analogous to bituminous mines. The low-volatile and high-volatile districts were divided by county units at the suggestion of George H.

Ashley, state geologist. The figures used are from Compensation Insurance Experience in Bituminous Mining in Pennsylvania, compiled by the Coal Mine Section of the Pennsylvania Compensation Rating and Inspection Bureau. This experience represents more than 50 per cent of the total tonnage in Pennsylvania from 1925 to 1928 inclusive, approximately 275,000,000 tons. Tabulations show accident severity rate, number of deaths with permanent totals and all other accidents, total cost, total days lost, cost per million tons and days lost per million tons. Other tabulations show this same experience by general causes of accidents and groups of seams of coal. Comparisons show there is little difference in the accident frequency between the low-volatile and high-volatile mines with the exception that the catastrophe experience shows a higher rate for high-volatile mines, as might be expected. Other conditions than the volatile matter of coal have been assigned as having a relation to the accident frequency. One of these is the height of the seam. The paper points out the fallacy of this idea. Geological conditions, pitch of seam and other natural conditions have also been assigned as having an effect on the accident frequency. It is shown that these have little influence and that the whole question of accident frequency is an operating problem and must be solved as such.

British Coal Industry Law Reviewed. By R. V. WHEELER. (*Preprint.* 2500 *words.*)—This paper is a brief discussion of the three first and most important parts of the British Coal Mines Act of 1930, namely Part I.—Production, Supply and Sale of Coal; Part II.—Reorganization of the Coal-mining Industry, and Part III.—Hours of Work. Anent Part I, the author, who is the head of the Mines Research Board of Great Britain, describes the regulation of output and prices and relates how this regulation is working out. An interesting feature is the transfer of quotas between coal-owners. Mr. Wheeler points out, apropos of Part II, that a Coal Mines Reorganization Commission has been appointed which considers that the vital need of the coal-mining industry is concentration of productive effort and regards consolidation as the only sure way of bringing about such a concentration. The provisions of the Act for hours of work are briefly stated under Part III.

Proposed Safety Code for Coal Mine Ventilation. (*Report of Subcommittee on Mine Ventilation Code of A. I. M. E. Committee on Ventilation.*)—The proposed code is the result of work started in 1928, from the previous foundation, leading up to the paper by G. S. Rice in February, 1928, and a paper by Graham Bright on Subjects Which May Be Considered by Subcommittees on Coal Mine Ventilation for Papers or for Encouragement of Research. From these papers, a questionnaire was formulated, the answers to which were used as the basis of the present proposed code. The aim has been to develop strictly an engineering code—one that shall be Utopian; something for an ultimate goal. A number of changes have been made in the original conception of the code, but at all times the scope of the work has been held to cover safety and health only, without regard to economic phases. The code as set up was not in its final state. Criticisms and suggestions were requested in time for discussion at the Bluefield meeting (October 9), so that points worthy of incorporation might be added before its next printing.

Petroleum and Gas

Stabilizing Influences. By EARL OLIVER. (*Trans., Petr. Dev. and Tech.,* 1931. 750 *words.*)—Introductory to Chap. I, Unit Operation. Proration, regulation of imports, unit operation, constructive marketing and interstate compacts all have their earnest proponents, but the advocates of each have been disposed to look with indifference on the other respective proposals, with the result that none has had united support. Proponents of these several remedies should determine its relative place in the major program of which it forms a part, how it should be coordinated

with other methods in that major program, and finally, the due process of law by which it should be established. The lawyer and the engineer must work closely together if the American petroleum industry is to recover from the economic ills that now burden it.

Compulsory Unit Operation of Oil Pools. By W. P. Z. German. (*Trans., Petr. Dev. and Tech.*, 1931. 8000 *words.*)—One meaning of the expression "unit operation of an oil pool" is that, without any merging of titles, the development and operation of the pool by the several proprietors shall conform, as nearly as practicable, to a plan which is intended to accomplish the greatest ultimate recovery of oil from the pool as a whole at a minimum of cost. The state cannot require a merging of titles, but it is believed that it can require unit operation as defined above, thus binding the lessors as well as the lessees. An ideal method of developing and operating a pool takes into account, among other things, the fact that the accumulated oil and gas in the reservoir is not evenly distributed therein; that said substances are originally held under a high pressure, called rock pressure; that, when property regulated and utilized, this pressure is of great value in the recovery of the oil; and that wells where gas only is found should not be operated, but only wells drilled into the oil-saturated portions of the formation. Present day proration ignores, while unit operation recognizes and enforces, an orderly development of a pool. The authority of a state to act to remedy the evils of waste, including underground waste, and disproportionate extraction rests upon its police power, a power to enforce the common-law maxim that no man may so use his own property as to injure the rights of his neighbors.

Unit operation by general agreement seems impossible of accomplishment. The migratory nature of the oil and gas deposits in pools under the lands of many proprietors; the value of a controlled utilization of the rock pressure in recovering the oil and the harm that can grow out of its dissipation and wastage by a few lessees; the desire of some lessees to obtain an advantage by excessive production; all these are facts which will justify the state in requiring unit operation in cases where it is not agreed upon. An ordinary conception of right and wrong should satisfy everyone that each land proprietor is entitled to a fair proportionate part, and no more, of the recovered oil and gas if at the same time he can be required to share on a like basis in the cost of its recovery. By the court decisions, any proprietor, unless and until restrained by legislative rule to the contrary, may extract and reduce to possession through his own wells, and thus own, all of the oil and gas that he is able. However, before production, each proprietor owns only that oil and gas which at the moment is under his land. The original right, quantitatively, of each proprietor, is measured by the oil and gas under his land before the pool is operated. If this right can be determined in advance by test wells, etc., then a basis is laid for a fair distribution of the oil that may be recovered from the pool regardless of the location of the wells through which it is produced. In 1900, the United States Supreme Court announced the law to be that, because of the peculiar nature of the right of each surface owner in a pool to reduce the oil and gas beneath to possession and the peculiar characteristics of the deposits themselves, their fugacious and migratory nature, it is within the power of a state to legislate for the protection of all of the collective owners by securing a just distribution to arise from the enjoyment by them of their privilege to reduce to possession. Decisions in several cases are cited: That compulsory unit operation would promote the prosperity of the landowners and their lessees in the saving of cost to the lessees and producing a greater ultimate net return on their investments and expenditures, and a more uniform and protracted income to the lessors, presents a circumstance favoring the validity of the requirement. The promotion of the prosperity of the people is within the police power. But the requirement, if made, would be best justified upon two sound propositions, first, that a state may by law conserve

this irreplaceable natural resource against waste, including underground waste, and may require that it be held in nature's storehouse until needed by the public; and, second, that a state may by law prevent any proprietor from so operating his property as to injure his neighbors.

Cooperation Between Engineers and Lawyers. By PETER Q. NYCE. (*Trans., Petr. Dev. and Tech.,* 1931. 2500 *words.*)—The present state of overproduction and demoralization in the petroleum industry has resulted from the early laws and court decisions holding that he who reduces oil to possession from a common pool becomes its absolute owner. The petroleum engineer has evolved principles and methods which make possible orderly drilling, the production of oil as the market demands, the elimination of waste, and the protection of property rights of all with injury to none. But these methods are difficult to apply because of the early decisions and the practices that have grown out of them. Subsequent laws and court decisions, however, are slowly adjusting themselves to the new methods. This tendency toward readjustment has been accomplished through education. Universal acceptance and successful operation of these better methods depend upon further education, not only of the oil and gas producers but of legislators and the public as well.

In our joint efforts to solve these problems we must recognize that the laws of nature are unchanging and unchangeable. They are inexorable; while man-made laws are properly subject to constant change as conditions change. They are merely rules by which society adjusts itself to conditions that prevail at the moment. The scientist and engineer strive to discover nature's laws; to explain them; to formulate them. The lawyer then must adjust human relationships to these natural laws so that nature's gifts shall become of practical value and use to mankind. It is, therefore, the function of the engineer to outline the conditions under which oil fields should be developed and of the lawyer to determine the method by which that can be accomplished legally.

Unit Operation in Oil Fields. By E. L. ESTABROOK. (*Trans., Petr. Dev. and Tech.,* 1931. 1100 *words.*)—Oil companies operating in foreign countries made increased use during 1930 of cooperative agreements in prospecting unproven territory and in developing proven territory. A considerable proportion of the existing foreign fields are held under single ownership and have been developed as units. (Paper contains some interesting detail on developments in the South Field, Mexico, and in Poland.)

Proration in Texas. By DAVID DONOGHUE. (*Trans., Petr. Dev. and Tech.,* 1931. 3000 *words.*)—In Texas the legal background of proration is the prevention of waste, the laws being administered by the Railroad Commission with the voluntary assistance of the Central Proration Committee. Periodical surveys are made as to supply and demand and submitted to the Commission in open hearings. The factors for successful proration plans are set forth. Stress should be put upon necessity of pipe line connection for each well. This in turn is assured by a low initial allowable for a field. As the market develops the allowable can be raised. The relative merits of proration methods in various districts are briefly discussed. A tabulation showing the proration methods in the various fields in Texas is included.

Proration of Yates Pool, Pecos County, Texas. By H. C. HARDISON. (*Trans., Petr. Dev. and Tech.,* 1931. 2000 *words.*)—The Yates pool, Pecos County, Texas, is the outstanding example of the benefits derived from the orderly development and operation of a field under effective proration. The perfection of the plan was the result of a year of earnest effort on the part of the operators to arrive at a formula that would express the relative value of the different leases and would apportion their outlet in conformity with these values. The field has produced 96½ million barrels of oil in a period of three years, under this plan, with little apparent decline in indicated poten-

tial production. The plan has lengthened the profitable life of the field and has resulted in the substitution of underground storage for tankage.

Unit Operation in California, with Discussion of Kettleman North Dome Association. By JOSEPH JENSEN. (*Trans., Petr. Dev. and Tech.*, 1931. 5500 *words.*)—California's outstanding contribution to unit operation is the plan of development established for the North dome of the Kettleman Hills. Under this plan the Kettleman North Dome Association will control the unified development and operation for 10,800 acres of proven and semiproven oil land, and at the same time the Standard Oil Co. will operate as another unit its fee land of 9460 acres. Holdings of the two companies are in a checkerboard mile-square pattern. No other major unit operation plans are under way in California at the present time. Areas that might be so considered are North Belridge, South Belridge, Lost Hills and Middle Dome of Kettleman Hills. The Kettleman Hills have long been regarded as a likely area for oil development. Discovery was made Oct. 5, 1928, by the Milham Exploration Co. Since that time the oil-bearing formation has been proved 1274 ft. thick and oil of 37° to 42° gravity has been produced in single wells at a 10,000-bbl. rate. Participation in the obligations and production of the Association is in proportion to the acreage held by each member. Likewise production obtained by the Kettleman North Dome Association unit and the Standard Oil Company unit will be based upon proportional acreage. Provisions for developing the true limits of the field are contained in the agreement.

Economic Aspects of Unit Operation of Oil Pools. By JOSEPH E. POGUE. (*Trans., Petr. Dev. and Tech.*, 1931. 2500 *words.*)—Unit operation is developing as a superior economic method of producing crude petroleum. It offers the means for reducing costs, eliminating waste and gaining the economic advantage of a *reserve without a potential.* The progress of unitization is being facilitated by the principle of differential cost, the principle of differential extraction, the drift toward large-scale operations, the need for rationalization of supply and the public interest in conservation. Progress in unitization is opposed by custom and the archaic law of oil and gas, which may be subject to change. Unitization offers the most promising means available for placing the production of crude petroleum on a sound economic basis.

Repressuring and Initial Pressuring. By H. C. GEORGE. (*Trans., Petr. Dev. and Tech.*, 1931. 1000 *words.*)—This paper presents the theory that it may be possible to flow wells in many new oil fields at no pressure lower than the initial rock pressure, during the economic life of the wells by building up a differential gas pressure above the initial rock pressure at key wells and introducing this gas compressed sufficiently above the initial rock pressure to induce flow towards the other wells, which are produced with a back-pressure equivalent to the initial rock pressure.

Problems in Proration on the Basis of Gas Energy. By EUGENE A. STEPHENSON. (*Trans., Petr. Dev. and Tech.*, 1931. 5500 *words.*)—It has been proposed by J. R. McWilliams that the energy of the gas within oil and gas pools might be used as a basis for proration. In order to bring about the successful application of such a plan it would be desirable to establish as early as possible the limits of the pool, the sand thickness in various localities, the porosity, gas pressures, and the areal extent of those portions of the field which carry free gas or which produce with high gas-oil ratios. Such information facilitates the calculation of the total gas volume and the total gas energy which can be made available. Three methods of estimating the gas reserves are illustrated, drawn from actual observations in various oil and gas fields. The advantages of the scheme proposed by McWilliams include a stimulus to more efficient production methods, the conservation of the gas pressure, the possibility of an orderly development program, lowered production costs, and higher ultimate recovery per acre. The recognition that the energy of the gas is the force which brings the oil

to the surface leads to the further conception that the entire pool may be legitimately regarded as the common property of the various operators.

Methods and Effects of Unit Repressuring in the Cook Pool. By GRAHAM P. CRUTCHFIELD. (*Trans., Petr. Dev. and Tech.*, 1931. 4500 *words.*)—The Cook pool, Shackelford County, Texas, unique because of the unity of ownership, has been continuously repressured since July 15, 1927. Residue gas and fractionator ends from the gasoline plant have been used for injection. A discussion is given of the practical problems encountered and solved in the design, operation and control of this unit. The results obtained are shown by a series of curves and progressive contour maps. These cover percentage of gas returned, rock pressures, oil production, gas-oil ratios, oil gravities, gasoline content of gas and ultimate recovery. The effect of proration on the curves is of special interest at this time. The very satisfactory increase (100 per cent) creditable to repressuring is shown by rate cumulative curves.

Development Program in a Part of the Ventura Avenue Oil Field. By JOSEPH JENSEN AND F. W. HERTEL. (*Trans., Petr. Dev. and Tech.*, 1931. 3500 *words.*)—A program of dividing the great thickness of oil-bearing formations in the Ventura Avenue field into zones has been instituted by the Associated Oil Co. in the eastern portion of the field. Areal extent of zones, position of edge waters and practical drilling depths have been used to divide this large thickness of oil formation into three zones, as this field does not contain distinctive shale bodies and intermediate waters separating the lower 3300 ft. of present known oil zone. There is no longer temptation to deepen wells for large production below when they are still producing commercial quantities of oil. The program is also designed to take care of development of deeper zones below the present known horizons.

Encroachment of Edge Water, Santa Fe Springs. By DONALD K. WEAVER. (*Trans., Petr. Dev. and Tech.*, 1931. 2500 *words.*)—The most prolific oil zones found in the Santa Fe Springs oil field are the Meyer zone, discovered in 1923 and occurring from 4100 to 4700 feet, and the O'Connell zone, first produced in 1929 and occurring from 6300 to 6900 feet. On top of the structure the lower O'Connell zone extends to a depth of 7150. Both the Meyer and O'Connell zones have had waters associated with them, with varying hydrostatic heads and salinities. While an original penetration of 600 ft. was possible on top of the structure within the Meyer zone, a low-head edge water gradually flooded the bottom 100 ft. of the zone between 1923 and 1925. It was necessary to plug off the bottoms of Meyer zone wells to about 4600 ft. In the closely drilled town-site area by 1927 water had reached a level of 4550 ft. and from the north an northwest sides of the field a higher water made its appearance about 330 ft. below the top of the zone. This water encroached along the bedding until it reached a level of about 4480 ft., where it flattened off in a plane. All waters in the Meyer zone show a salinity ranging from 450 to 675 grains per U. S. gallon. Very successful plugging has been possible, in almost each case resulting in total elimination of the salt water and increases in daily net oil production of from 50 to 250 bbl. Three waters have been identified in the O'Connell zone, the upper two being low-head intermediate edge waters and the other being high-head bottom water occurring on the flanks of the structure below about 6925 ft. Only the area within the two top closing contours of the field is at present free from the two intermediate waters, the upper of which occurs 150 ft. below the top of the zone and the second from 310 to 320 ft. below the top. In the first two years of the producing life of the O'Connell zone these two intermediate waters have encroached to a point about 75 ft. structurally higher than that at which they were originally found, and from 600 to 700 ft. horizontal distance. No attempts have been made so far to plug off these intermediate waters and produce above and below them. Wells are either produced wet, or the entire lower portion of the zone sacrificed. While practically all wells in the field were bottomed shallow enough to avoid the high-head bottom water, this water

is acting as a water-drive, causing the lowest portion of the O'Connell zone to be very prolific. When this water is present the head and volume is sufficient to prohibit any entry of oil from the productive sands above it.

Water Problems of the McKittrick Oil Field. By JOSEPH JENSEN AND J. B. STEVENS. (*Trans., Petr. Dev. and Tech.*, 1931. 1800 *words.*)—The McKittrick oil field, Kern Co., California, was discovered in 1898 and has had a production to date of 95,000 bbl. per acre. The structure is an accumulation against overthrust faulting. An adjacent basin at higher elevation supplied the source of a fresh and sulfur water having access to the field as top and intermediate waters. During the first 20 years the field developed a water problem that threatened to destroy it. Heavy pumping of fresh water in the basin for domestic and field use during the succeeding 10 years lowered its water table 100 ft. This with heavy pumping in the oil field lowered the fluid level in the wells over 200 ft. and practically exhausted the top and intermediate waters. The future of the field now depends on the behavior of the bottom and edge salt water.

Bottom-hole Pressures in Oil Wells. By C. V. MILLIKAN AND CARROLL V. SIDWELL. (*Trans., Petr. Dev. and Tech.*, 1931. 5300 *words.*)—Production control and lifting procedure can be more intelligently directed when bottom-hole pressures and pressures within the producing formation are known. The pressure data presented were determined with an Amerada recording pressure gage which has an outside diameter of 2 in. It has been determined that the variation of pressures within different formations relatively close together may be fairly large. By comparing the pressures at the bottom of the well when closed in and while producing, the operator may determine whether the rate of production being obtained is limited by the capacity of the method of lifting the oil or by the capacity of the well to produce. The best size of flow string for a well flowing naturally or by gas-lift must be determined by the use of an estimated, calculated or measured bottom-hole pressure, and the degree of accuracy is in proportion to the accuracy of the bottom-hole pressure upon which the calculation is based. Production control used to obtain more efficient use of the gas energy accompanying the oil, to retard bottom water invasion, or to obtain more effective natural water flood, is usually accomplished by regulating the pressure at the casinghead or changing the operating method. These are indirect methods because a change in the rate of production is a result of change in the bottom-hole pressure (more specifically a change in the differential pressure between the producing formation and the bottom of the hole) caused by a change of the casinghead pressure or method of operation. Reliable pressures at the bottom of oil wells and in the producing formation are essential in solving problems of lifting and recovery of oil.

Bottom-hole Beans—Theory, Methods and Effects of Their Use. By WILLIAM A. CLARK. (*Trans., Petr. Dev. and Tech.*, 1931. 3500 *words.*)—Bottom-hole beans in various forms have been used, more or less haphazardly, for some time. Only in the last year or two has there been any scientific experimentation with this method of producing wells. In theory, the use of bottom beans utilizes more efficiently the energy of expanding gas in a flow string; prevents surging flow and slippage of gas; effects the positive control of gas-lift wells; and probably helps to prevent edge water coning and sand troubles. The relation of areas of bottom to surface beans for any one well may be roughly calculated. This paper describes and compares a few types of beans in each of three classes: the fixed class, the adjustable class and the replaceable class. From the records and comparative data presented the use of the beans apparently reduces gas production in relation of that of oil, effects a more sustained rate of flow, and, particularly with gas lift wells, prevents heading and allows definite control of production and more economical operation.

Density of Oil-gas Columns from Well Data. By WM. VICTOR VIETTI. (*Trans., Petr. Dev. and Tech.*, 1931. 2000 *words.*)—Estimation of the density of columns of

oil and gas under actual flowing conditions is performed by extrapolating pressures back to zero flow. The zero flow pressure thus obtained is subtracted from the known bottom-hole closed-in pressure to calculate the density of the column. Data from several wells give the change in density with change in gas-oil ratio. Curves and tables are included to illustrate the method followed.

Characteristics of Drilling Fluids. By C. P. Parsons. (*Trans., Petr. Dev. and Tech.*, 1931. 2800 *words*.)—The characteristics of drilling fluids are dependent upon the proportions and activities of the various solid constituents that are present in three states of division; namely, the suspended state, the colloidal state, and the crystalloidal state in which the particles are held in true solution. Sand, gravel and cuttings are examples of suspensoids and contribute nothing to the desirable properties of a drilling fluid. The colloidal particles are those which actually do the useful work of a drilling fluid. However, much of the stability of the colloidal particles is dependent on the electrolytic condition of the crystalloidal particles. A stable colloidal solution as applied to drilling fluids consists of very fine solid particles kept from settling by a protecting film or agent, usually either protective colloids or electrolytic action or both. Such properties present possibilities for chemical treatment of rotary drilling fluids whereby thickening or thinning may be obtained by the proper selection of inexpensive chemical reagents, without affecting the weight. Theoretically, clay particles reach a state of maximum dispersion in a fluid when alkali is added in sufficient quantities to raise the pH value to above 9. Actually, however, in many drilling fluids, the reaction is entirely different. In such cases the addition of alkalies or alkaline salts produces flocculation and increased viscosity. This effect is due to the presence of soluble metallic salts in the fluid. The addition of alkali causes these metallic salts to form gelatinous precipitates which increases the viscosity of the mud.

Preliminary Report on an Investigation of the Bureau of Mines Regarding the Solubility of Natural Gas in Crude Oil. By Ben E. Lindsly. (*Trans., Petr. Dev. and Tech.*, 1931. 10,000 *words*.)—High-pressure samples of oil were obtained at 70° F. from the well head of flowing wells, and the dissolved gas was released, measured and analyzed. The results differed materially from the results reported by previous investigators, who determined solubility by contacting oil that had been in storage for some time, with dry natural gas. This difference is shown by comparison with a sample of Kettleman Hills crude of 44.5° A.P.I. gravity. This oil liberated 275 cu. ft. of gas per barrel of oil when reduced from 300 lb. per sq. in. to atmospheric pressure, whereas previous reports showed that only 70 cu. ft. of gas was absorbed by Bradford crude of 44.3° A.P.I. gravity over the same pressure range. The difference of 205 cu. ft. of gas is due mainly to the presence of large quantities of the heavier gases that occur naturally in the Kettleman Hills and other crude oils, but which ordinarily do not remain in large proportions in storage oil. The solubility curves do not show the straight line relationship in accordance with Henry's law. One sample showed more than three times the amount of dissolved gas in the lower pressure ranges than it would have contained if the straight line relationship had held. Henry's law holds fairly well when applied to a single gas and a single solvent, but not to the liberation of gas naturally in solution in crude oil. It was shown also that the amount of shrinkage caused by the liberation of dissolved gases varies with the character of the oil. A high-gravity crude showed a shrinkage of 36 per cent in liberating 800 cu. ft. of gas per barrel of oil, whereas with other oils of lower gravity, the shrinkage may be less than 10 per cent. The shrinkage curves are not straight lines but show increasingly greater shrinkages in the lower pressure ranges, where the gases which are liberated are much heavier than in the upper pressure ranges.

Experimental Measurement of Slippage in Flow through Vertical Pipes. By T. V. Moore and H. D. Wilde, Jr. (*Trans., Petr. Dev. and Tech.*, 1931. 8500 *words*.)—This paper presents the results of theoretical study and experimental

investigation of the problem of the flow of mixtures of oil and gas in vertical pipes. From theoretical considerations, it is shown that the fraction of the total energy of expansion of the gas which is lost in slippage is equal to the ratio of the relative velocity of the gas and liquid to the actual linear velocity of the gas. In the experimental work, provision was made to determine relative velocities, as well as linear velocities of the liquid and gas, by measuring the fraction of a vertical pipe which was filled with liquid or gas under various conditions and rates of flow. An equation is presented to show approximately the relationship between this fraction and other variables more easily measured. An approximate expression for friction losses is also given. These formulas apply satisfactorily to short gas-lifts, but can be used only as approximations in calculations on long lifts or actual wells.

Practical Interpretation of Core Analysis. By L. S. PANYITY. (*Trans., Petr. Dev. and Tech.*, 1931. 4000 *words.*)—The purpose of this paper is to indicate the value of screen analysis, especially in areas where repressuring by means of water-flood, air or gas is being used in the partly depleted fields. Sand samples have been analyzed for porosity, saturation and grain size. They are mainly from the Bradford water-flood area, where a great reliance has been put upon such analyses, and where many data are available. A few sand studies are included from fields other than the water-flood area, in order to indicate the possibility of screen analysis in any field. It appears reasonable to the writer to expect that the possibilities of water-flooding in certain parts of the Bradford field, and in many of the older fields of the country where difficulties were encountered, would have been much better understood if there had been a thorough sand analysis along the lines laid down in this paper. He believes that the size of the individual pore space, and not porosity, is the criterion for water-flooding, and that it is the controlling factor.

Permeability Studies of Pennsylvania Oil Sands. By CHARLES R. FETTKE AND W. A. COPELAND. (*Trans., Petr. Dev. and Tech.*, 1931. 3300 *words.*)—Permeability tests with both air and water on 18 samples of typical Pennsylvania oil sands are described. Curves showing the relation of permeability to porosity, plotted on semilogarithmic graphs, indicate that in most instances porosity is a more important factor in determining relative permeabilities than actual grain size in sandstones of the type most frequently found in the oil and gas fields of Pennsylvania, but grain size must also be taken into consideration. Grain size becomes an increasingly important factor in determining permeability as the porosity increases above 17 per cent.

Microscopic Study of California Oil-field Emulsions and Some Notes on the Effects of Superimposed Electrical Fields. By HARMON F. FISHER. (*Trans., Petr. Dev. and Tech.*, 1931. 8000 *words.*)—A brief description is given of the special microscope technique developed for a rapid comparative examination of the general characteristics of crude-oil emulsions, of the nature of their electrical charge and of the effect of a superimposed electrical field; generalized procedure for making photomicrographs is also outlined and the reasons for certain precautions are explained. This is followed by photomicrographs with a brief description of six typical California fresh production emulsions, four typical California tank bottoms and sump oils, including an interesting view of an "emulsifying" film and for comparative purposes, four miscellaneous samples from sources outside of California depicting double emulsions and oil-in-water emulsions. A series of motion picture photomicrograph studies is included showing the course of a cataphoresis tests on a typical California emulsion; also a motion picture photomicrograph study of an electrical dehydration experiment conducted under the microscope.

Oil and Gas Production in the Eastern District during 1930. By J. FRENCH ROBINSON. (*Trans., Petr. Dev. and Tech.*, 1931. 3500 *words.*)—The development of a field in Chautauqua and Cattaraugus counties, New York, has been rapid and probably reached its peak of production in 1930. The gas in this field comes from the White

Medina Sand (Silurian), at an average depth of 2600 ft. Some of the wells had an open flow of 20,000,000 cu. ft. of gas per day, with an original rock pressure of 965 lb. per sq. in. Another area developed during 1930 was the Tyrone field, in Tyrone Township, Schuyler County, New York. This field was discovered and is practically controlled by the Belmont Quadrangle Drilling Corpn. of New York. The production is from the Oriskany Sand (Devonian), which is found at an average depth of 2100 ft. The field covers approximately 2000 acres, and to date has 14 producing gas wells. The largest well had an open flow of 15,000,000 cu. ft. and an original rock pressure of 740 lb. The gas from this field is impregnated with sulfur, and requires washing before it is marketed. The Oriskany sand outcrops at Oriskany Falls, N. Y., and has a thickness of 10 to 12 ft. Its texture is coarse grained, it is composed mostly of quartz and is a regularly bedded sandstone. At Yauger's Wood, in Cayuga County, New York, it is exposed and shows a thickness of 26 feet. The discovery of the Tyrone field led to much prospecting in New York and the northern part of Pennsylvania, and as a result of this search, the Allegany Gas Co. struck a large gas well in Farmington Township, Tioga County, Pennsylvania, which created considerable excitement through the Eastern District. The gas was encountered at a depth of 4010 ft., and tested 20,000,000 cu. ft. open flow, with a shut-in pressure of 1750 lb. It is generally supposed that the producing horizon is the Oriskany but this has not been definitely determined. Twenty or more of the larger companies, besides many independent operators, are planning development in this field. Approximately 25 interesting tests are under way now, the results of which will be eagerly watched. Considerable drilling has been done in Clearfield County, Pennsylvania, with varied results. Several dry holes have been drilled near two good gas wells. The wells produced from the Sheffield sand, and had an open flow of approximately 2,000,000 cu. ft. each. A 750,000-cu. ft. gas well was completed in Quemahoning Township, Somerset County, Pennsylvania. The gas was struck at a depth of 2400 ft., and had a rock pressure of 800 lb. In West Virginia, gas was struck in Monroe County, but the production from this county to date does not warrant an extensive drilling campaign. In Horse Cave field in Hart County, Kentucky, a 10,000,000-cu. ft. well was struck at a depth of 316 ft. This gas was found in a formation called the Black Lime. Approximately 50 wells are drilling in this locality at the present time.

Petroleum Production in Middle Western States, 1930. By R. B. NEWCOMBE. (*Trans., Petr. Dev.* and *Tech.*, 1931. 3000 *words.*)—Despite abnormal economic conditions, total production of oil in the Middle Western States did not greatly decline during 1930. The output for Illinois, Indiana, Kentucky, Michigan and Ohio was slightly under 25,000,000 bbl. Kentucky was the leading state of the group with a little over 7,500,000 bbl. The Legrande oil pool in Hart County, Kentucky, offered the most sensational oil production, and several important extensions were discovered in the central Michigan area. Michigan also offered potential possibilities for commercial development of natural gas in the Broomfield pool west of Mount Pleasant, Isabella County. Considerable activity centered about the exploiting of natural gas resources in Kentucky. Output of oil in Kentucky was voluntarily prorated by the operators. The market outlet at Toledo was increased by the completion of the Pure Oil Co. refinery and acquisition of the Paragon refinery by the Gulf Companies. Pipe line facilities were augmented in several of the Middle Western States.

Oil and Gas Development in South Arkansas, North Louisiana and Mississippi, 1928 to 1930. By H. K. SHEARER. (*Trans., Petr. Dev. and Tech.*, 1931. 5000 *words.*)—The principal event in north Louisiana during 1928, 1929 and 1930 was the discovery and development of the Zwolle oil field, which caused a small increase in total oil production in 1930. Two new Lower Cretaceous gas fields, at Sugar Creek and Rodessa, were discovered. There has been extensive development of the Monroe and Richland gas fields, following the completion of pipe lines to Atlanta, Birmingham,

Memphis, St. Louis and New Orleans. In south Arkansas drilling and production have declined steadily, although the Champagnolle field was developed principally in 1928 and the small Urbana field was discovered in 1930. These fields, producing from the Lower Cretaceous Trinity red beds, indicate possibilities of future deep production. The first important gas field in Mississippi was discovered at Jackson in 1930. Production comes from Upper Cretaceous chalk, on a pronounced structural high, which is reflected in the surface Eocene and Oligocene formations.

Production and Development Situation of Kansas in 1930. By HENRY A. LEY. (*Trans., Petr. Dev. and Tech.*, 1931. 7500 *words.*)—Kansas ranked fourth in the list of oil-producing states in 1930. This paper contains much detailed discussion and statistics on the situation in Kansas in 1930 as relates to production and development, and new discoveries of petroleum and natural gas.

Production in Oklahoma during 1930. By HENRY A. LEY. (*Trans., Petr. Dev. and Tech.*, 1931. 6000 *words.*)—Oklahoma produced about 40,000,000 bbl. less crude oil in 1930 than it did in 1929, but developed the largest initial production from current well completions ever recorded in its history. The output would have exceeded all previous years had it not been for state-wide production curtailment. Much detail and statistics on developments and production and new discoveries are given, and two pages are devoted to the natural gas situation.

Petroleum Developments in Texas During 1930, Except the Gulf Coast and Panhandle Districts. By M. G. CHENEY. (*Trans., Petr. Dev. and Tech.*, 1931. 3500 *words.*)—The main problem of the year was the equitable reduction of production to meet decreasing demand for oil. Supplementing decreased drilling, proration of output by order of a state commission was attempted in August, 1930, unde conservation laws forbidding physical waste. Injunctions permitted successful evasion, but most producers submitted willingly to a percentage outlet according to the needs of the pipe line system to which connection had been made. These needs varied and even vanished in some cases and distress selling of crude developed which gradually undermined market prices. More effective conservation laws to restrain all producers equally, and more cooperation by purchasers in equalizing outlet, are needed if production is to be kept in balance with demand, which is the sound economic basis of proration. Reduced drilling and "statewide" proration by purchasers led to reduction in daily production. Deep sands of West Texas and shallow sands of North Central Texas contributed outstanding developments during 1930. There was rapid competitive development of the Darst Creek pool of Southwest Texas and the orderly development of the unitized Van pool of Northeast Texas. Several promising structures in the Van district failed to produce, but wildcat tests 50 miles to the east found the eastern margin of the Woodbine sands and as the year closed development in western Rusk and Gregg counties overshadowed all else in the Mid-Continent oil fields and threatened the industry with chaos.

The fictitiousness of certain pool potentials was shown by Darst Creek and Hendricks, rated respectively at 180,000 and 1,000,000 bbl. daily but actually yielding but 78,000 and 52,000 respectively when opened to continuous full production. The Yates pool showed gradual encroachment of water when produced above 120,000 bbl. daily, but reduction to 91,000 bbl. daily near the close of the year seemed to eliminate this hazard, for the present. Thus in effect West Texas has lost several million barrels daily potential during 1930.

Petroleum Developments in Texas Panhandle in 1930. By W. E. HUBBARD AND H. E. CRUM. (*Trans., Petr. Dev. and Tech.*, 1931. 3500 *words.*)—The Panhandle of Texas produced 33,000,000 bbl. of oil in 1930, or about 9 per cent more than in 1929, bringing the grand total for the area up to 157,000,000 bbl. The present production is slightly over 40,000 bbl. under state-wide proration, although the daily potential of the area is well over 100,000 bbl. Of the 526 completions for the year, 308 were

oil wells, 169 produced gas and 49 were dry; 36 oil wells were plugged and abandoned while 15 were plugged back for gas; 4,139 million feet of open flow were added by the completion of 169 gas wells. Fifty-three casinghead gasoline plants produced 7,240,000 bbl. of gasoline and 24 carbon-black plants produced 110,000 tons of carbon black. The outstanding developments were the completion of a 600-bbl. well in Moore County, the bringing in of a 1900-bbl. wildcat on the south side of the Arch in Carson County, the extension of the Bowers and Finley pools in Gray County, the completion of a 12,000-bbl. semiwildcat well a few miles southwest of Pampa in Gray County, and the rather rapid development of the dolomite production in northwestern Gray County.

Petroleum Developments on the Gulf Coast of Texas and Louisiana during 1930. By L. P. Teas. (*Trans., Petr. Dev. and Tech.*, 1931. 4000 *words.*)—The year 1930 may be regarded as an average year for the Gulf Coast. During the year two additional salt domes and seven structural areas were proved productive. Six fields on the Texas Gulf Coast and five fields on the Louisiana coast reached production peaks in 1930. Geophysical activities waned and deep drilling became increasingly popular. Nineteen wells were producing from below 6000 ft. at the end of the year compared with but one well at this depth at the beginning of the year. On five domes new producing areas were discovered.

Petroleum and Natural Gas Development in the Rocky Mountain District, 1930. By R. Clare Coffin. (*Trans., Petr. Dev. and Tech.*, 1931. 5800 *words.*)—The production of petroleum in this region increased from 27,104,436 bbl. in 1929 to 33,048,630 bbl. in 1930. Three new light-oil fields were discovered, but the oil reserves suggested by the deepening of wells in existing fields during 1930 are probably of greater extent. The development of the Hobbs oil field in New Mexico was the outstanding feature in the region.

Petroleum Developments in California during 1930. By B. E. Parsons. (*Trans., Petr. Dev. and Tech.*, 1931. 7000 *words.*)—At the beginning of 1930, with drilling operations only slightly checked, the California industry faced a steadily increasing potential production, an actual production in excess of market requirements and stocks of crude oil and refined products which had reached staggering proportions. During the year, the intensive development of the Playa Del Rey field and extensions to several oil fields, together with flush production from numerous large wells from both new and old fields caused a steady increase in the state's potential production to a record peak in November of 1,186,000 bbl. daily. Despite this burden and the handicap of incomplete cooperation in several fields, curtailment successfully lowered the trend of crude oil production in California, except for the period of an open-flow test to determine actual potential production in several important fields in February. Under the curtailment programs in effect, the state's production of crude oil was reduced 22 per cent under 1929 and even slightly under the production for 1927 and 1928. The total stocks of crude and refined products were reduced 2.7 per cent during 1930, as compared to an increase of 30 per cent in total stocks of all products during 1929. Drilling operations were reduced 28 per cent as against 1929.

Petroleum and Natural Gas in Canada during 1930. By Linn M. Farish. (*Trans., Petr. Dev. and Tech.*, 1931. 2000 *words.*)—The reported production of petroleum in the Dominion of Canada for 1930 was 1,555,199 bbl., an increase of 458,373 bbl., over 1929. The Province of Alberta, largely due to the Turner Valley field, accounted for nearly all the production with Ontario and New Brunswick, the other two productive provinces, contributing less than 1 per cent of the total. The consumption of natural gas for all purposes in Canada reached a new record of practically 40,000,000,000 cu. ft. in 1930, which was a considerable increase over 1929. The estimated value of the gas output was $10,500,000, against a little over $5,000,000 for the total petroleum production. Gas wells in Alberta accounted for 75 per cent of the total,

with Ontario second and New Brunswick a poor third. Very little change in the production of either gas or oil is anticipated in Canada during 1931.

Mexican Oil Fields during 1930. By V. R. GARFIAS AND C. O. ISAKSON. (*Trans., Petr. Dev. and Tech.*, 1931. 4500 *words.*)—Mexican production continued to decline during 1930, and Mexico has fallen to fifth or sixth place among oil-producing countries. The year 1931 should show a continued decline in the fields tributary to Tampico, but the fields on the Isthmus of Tehuantepec should maintain their production. A pipe line is being built from Palma Sola to a new refinery being constructed in Mexico City. A number of wildcats are being drilled in northeastern Mexico. Production for 1931 should be about 35,000,000 bbl., unless curtailed by proration or import regulations.

Petroleum Developments in the Argentine in 1930. By GILBERT P. MOORE. (*Trans., Petr. Dev. and Tech.*, 1931. 3500 *words.*)—Argentine production in 1930 amounted to 8,909,773 bbl., a decrease of 481,613 bbl. from the 1929 production. The Comodoro Rivadavia fields were down about 850,000 bbl. below 1929 figures but other fields in Neuquen and Salta increased their production considerably. Salt water is coming into the Government-operated field at Comodoro in increasing quantity. The Government is developing a new field at Canadon Perdido, west of Comodoro, which appears to have great promise. In the Neuquen district most of the oil is coming from the Dadin field. The wells on the old Government reserve are said to be dwindling in production very rapidly. No new fields have been discovered in this area although geophysical work is now being carried out in search for new drilling locations. In the Salta district the Lomitas field and Campo Vespuccio are being developed. The Government plans to drill three wildcat wells in the Salta district. It is reported that the Lomitas field will be electrified in the near future. The Royal Dutch Shell interests have obtained permission for the erection of a refinery at Buenos Aires which they expect to supply with their own production from Comodoro Rivadavia. Argentina produces about 40 per cent of its requirements of petroleum products and has a constantly increasing consumption.

Petroleum Developments in Bolivia and Chile in 1930. By GILBERT P. MOORE. (*Trans., Petr. Dev. and Tech.*, 1931. 500 *words.*)—Petroleum operations in Bolivia are still limited to those of the subsidiaries of the Standard Oil Co. of New Jersey. Two completions were reported during the year, one of which was a failure. It is reported that Bolivian oil will be exported and sold during 1931. A new petroleum law was passed and later cancelled during 1930. In Chile the Belgian company, which is drilling two wells for the Government near San Antonio, in southern Chile, has made very little progress. The Government has a staff of German geophysicists working in the Magallanes district. A refining monoply was decreed during the year, but existing companies are permitted to continue operations.

Oil Development in Peru in 1930. By OLIVER B. HOPKINS. (*Trans., Petr. Dev. and Tech.*, 1931. 900 *words.*)—Activity was confined almost entirely to the three old producing fields in the northern coastal area. In the eastern part, in the valleys of the Maranon, Ucayali and Huallaga rivers, additional geologic work was carried on, but no drilling was undertaken. The Government granted a concession to W. R. Davis and associates for the building of a railroad across the Andes, which carries with it a grant to the concession of 12,500,000 acres of land with oil and mineral rights. Peruvian production was cut 8 per cent in an effort to conform with the available market.

Petroleum Activities in Colombia during 1930. By JAMES TERRY DUCE. (*Trans., Petr. Dev. and Tech.*, 1931. 1200 *words.*)—There were no important changes in the producing situation in Colombia in 1930, because the emergency bill passed by the Colombian Congress in 1927 was still in effect, and therefore no change had taken place in the law governing petroleum. Production in Colombia will depend

upon legislation and improvement in the general economic conditions affecting the oil industry.

Petroleum Development in Venezuela during 1930. By C. W. HAMILTON. (*Trans., Petr. Dev. and Tech.*, 1931. 4000 *words.*)—A brief review of 1930 petroleum development in Venezuela. Production for the year was about the same as for the previous year. Drilling declined sharply. Exports somewhat increased but only by about the same amount as the decrease in crude stocks in field storage. Because producing fields are rapidly reaching a settled stage, gas lift and pumping operations were extended by all companies. Wildcat exploration in the Maracaibo Basin area was active but not very successful. Present indications suggest that further decline in production, shipments and drilling will occur in Venezuela during 1931.

Trinidad Oil Fields in 1930. By WILLIAM J. MILLARD. (*Trans., Petr. Dev. and Tech.*, 1931. 2500 *words.*)—The oil of Trinidad comes principally from 3100 ft. of Middle Miocene, although there is a total of 5600 ft. at least of Tertiary oil-bearing serata. Production is confined largely to the southwest portion of the island. Actual production from strong folded sediments ranges from 4000 to 18,000 bbl. per acre. Production for 1930 is estimated at 9,500,000 bbl. About eight companies operate in Trinidad of whom three are American. Because of different fineness of oil sands two methods of drilling the sands by rotary have developed; *i.e.*, the use of perforated pipe and the use of screened pipe. Both, however, use a 2-in. wash pipe held by cast-iron rings. Leases range from 5 to 5000 acres. Rentals range from $5 to $15 per acre. Future reserves may total more than one billion barrels.

Russian Oil Fields in 1929–1930. By BASIL B. ZAVOICO. (*Trans., Petr. Dev. and Tech.*, 1931. 5000 *words.*)—The oil industry in the Soviet Union closed the 1929–1930 operating year fulfilling its assigned program. During this period of time, however, no basic improvements were noted within the industry. A careful study of the conditions indicates that disorganization in most branches increased continually; that supply and transportation problems are approaching a state of breakdown; that labor problems become increasingly difficult and that the estimates for the 1931–1933 period are too large. The production of all Russian fields increased by 27,200,000 bbl., as compared with the 1928–1929 operating year, while the next year's program anticipates a further increase of 46,000,000 bbl., and the 1933 plans call for an amazing total of 330,000,000 bbl., or over 150 per cent more than the production of the operating year that has just closed.

Petroleum Development in Rumania during 1930. By IONEL I. GARDESCU. (*Trans., Petr. Dev. and Tech.*, 1931. 2800 *words.*)—Production of oil was 42,547,000 bbl., an increase of 19 per cent over the previous year. Deep drilling in Moreni was chiefly responsible for the increase in production, which reached a high mark at 128,000 bbl. per day. A temporary agreement of curtailment was reached during the latter part of the year. The Baldesti gas field (dry methane gas) proved to be oil-bearing in a deeper horizon and promises to be one of the largest producing oil fields of Rumania.

Petroleum in the Dutch East Indies. By FRED B. ELY. (*Trans., Petr. Dev. and Tech.*, 1931. 4300 *words.*)—Production in the Dutch East Indies for the year 1930 is given as 39,873,848 bbl. and in Sarawak as 5,832,000 bbl., making a total of 45,705,848 bbl. against a total of 43,201,918 bbl. for the year 1929. The marked increase for 1926 to 1930 inclusive is due principally to the bringing into production of the Talang Akar field of the N.K.P.M. (subsidiary of Standard Oil Co. of New Jersey). It is estimated that this field will produce close to 16,000 bbl. per day during 1931. Other than the increase in production from the Talang Akar field and possibly one or two other fields which may be brought in on concessions already controlled by the two major operating companies, it is not anticipated that the yearly increase in production will be very marked.

Petroleum in the Indian Empire. By Eric J. Bradshaw. (*Trans., Petr. Dev. and Tech.*, 1931. 7500 *words.*)—For the past 10 years India's total production of petroleum has remained practically constant and is now relatively unimportant. A slow decline may be anticipated in the future and there is but slight prospect of the discovery of new major fields. The bulk of the total supply comes from the Yenangyaung and Singu fields in Burma, the crudes resembling those of the Texas Panhandle with an average A.P.I. gravity of 36.4. The indigenous supply of petroleum is now insufficient to meet local demands and the only export of importance is paraffin wax. There are three main petroliferous areas in the Indian Empire: the Burmese Gulf, the Assam Gulf and the Punjab-Baluchistan Gulf. In each the oil is associated with Tertiary strata and is found in anticlinal structures. The more important fields in each area are described separately, details being given of the mode of occurrence of the oil and the methods of production and development adopted or contemplated. The total annual production from each is listed for the past 10 years.

Petroleum Development in Africa. By W. B. Heroy. (*Trans., Petr. Dev. and Tech.*, 1931. 600 *words.*)—Commercial production in 1930 was restricted to two countries: Egypt, 277,146 metric tons, and Algiers, probably less than 1000 metric tons. Active prospecting for petroleum was under way in Egypt, Mozambique, Angola and Morocco.

Interest Rates and the Oil Industry. By Barnabas Bryan, Jr. (*Trans., Petr. Dev. and Tech.*, 1931. 1900 *words.*)—Had the oil industry heeded the call money signal in July, 1928, and ceased all unnecessary fixation of money in physical property, the present outlook would be one of great prosperity. There would be no great potential production and little need for proration. There would not be such a large surplus of cracking capacity. Many retail stations with their background of trucks, tanks and tank cars would not be in existence. Since there is no probability that industry will ever correct the business cycle by preventing the credit climax, there is a great opportunity for any company that can develop the mental balance to deflate its position as credit tightens and expand to the fullest possible extent of inflation in the depth of national depression.

Gasoline, Its Relation to Petroleum Economics. By H. J. Struth. (*Trans., Petr. Dev. and Tech.*, 1931. 4500 *words.*)—That there is a direct relationship between the financial results and the ratio of demand to supply of gasoline was demonstrated in the facts presented in this paper, which clearly shows that the degree of prosperity or depression accompanying the oil industry's developments is to a large extent influenced by the degree of balance maintained between the supply and demand of gasoline. The results of the year 1930 are cited as indicative of the folly of maintaining gasoline production in excess of normal market requirements, showing that top-heavy inventories of gasoline undermined not only the refined oil market but the crude market as well. He stated that burdensome gasoline stocks really defeated the realization of benefits that might have accrued to the industry from an otherwise favorable statistical position in the producing branch. A series of graphic charts presents facts showing the relative effect of economic and uneconomic refining operations in 1930; percentage division of a barrel of crude in products in volume and value; trend of gasoline prices in relation to the ratio of demand to supply; normal course of crude runs during 1931; theoretical normal course of gasoline stocks compared with actual course since 1918; annual combined net profits of ten integrated oil companies from 1925 to 1930, inclusive. The concluding remarks point out that there is a distinct opportunity for the petroleum industry to effect a degree of stabilization that will invite more prosperous times. That opportunity lies in a program of production control built upon a foundation of a balanced gasoline supply.

Economics of Distribution in the Oil Industry. By Sidney A. Swensrud. (*Trans., Petr. Dev. and Tech.*, 1931. 5500 *words.*)—The oil industry today is essentially

the gasoline industry. The demand for gasoline developed rapidly, but at the start marketing and crude oil production were separate activities. Integration rapidly took place, however, and today most major companies engage in all activities of the business from producing crude oil to marketing finished products, providing a direct channel for the exertion of crude oil pressure on the marketing of gasoline. The estimates of a decade ago predicting the early exhaustion of crude proved completely inaccurate and instead there has been a constant overproduction of crude oil since 1921. This factor has profoundly affected the marketing picture because large companies owning crude oil have been forced to seek outlets for it in the form of gasoline, thus promoting a tremendous overexpansion of marketing facilities, which was aggravated also by wide margins between the wholesale cost and retail price maintained during the period of most rapid increase in demand, encouraging uneconomic jobbers and dealers to go into the business. Wide margins to dealers have promoted price-cutting on their part, and certain new forms of outlets, such as the tank-car service station, tend to cut price habitually, thus reducing the total margin between cost and retail. At the same time, oil companies, pressed for sales, are finding it necessary to spend more money advertising, increasing quality, and providing more expensive marketing facilities. The number of outlets is now increasing faster than gasoline consumption, causing continual dilution in the average gallonage per outlet, thus further increasing marketing expense per gallon concurrently with declining margins. These factors, however, have not yet begun to cause a slowing down of marketing expansion, because of the tremendous pressure of crude oil. It is probable, therefore, that many oil companies are spending much of the revenue from other ends of their businesses in the building up of high-cost distribution, and it is believed that the final struggle for the survival of the fittest will focus most sharply in the marketing field.

Stabilization of the Petroleum Industry. By LEONARD LOGAN. (*Trans., Petr. Dev. and Tech.*, 1931. 3800 *words.*)—The petroleum industry, along with other primary industries, has suffered during a longer period of depression than have the secondary industries. This industry may trace its troubles, other than those due to cyclical influences, to the institution of property rights and the migratory nature of oil. This paper deals with only the latter influences. The institution of property rights and the migratory nature of oil through the operation of competitive forces have brought about a serious condition of overproduction. The three major plans proposed for the curtailment of the supply are unit operation, proration, and the tariff. Unit operation was not discussed by the writer because it was treated in other papers. However, the writer thinks this the best plan. The tariff would be ineffective because the industry is on an exporting basis. Proration was considered more in detail. Proration is only a temporary expedient and would not be suitable to follow as a long-time policy. A commission representing the independents, the larger integrated companies and the Government is suggested, but this plan is not likely to receive approval by the industry at large. Unless the conflicting interests within the industry get together on a cooperative basis there is little hope for stability of the industry.

Production Cost as a Factor in Oil Economics. By H. J. WASSON AND LUCIUS W. MAYER. (*Trans., Petr. Dev. and Tech.*, 1931. 3000 *words.*)—Whether the economics of oil have ever been fundamentally sound is a question. The oil-scarcity delusion caused the establishment of oil prices and land values on unsound principles, because of a false sense of profit security. Value has a close relation to the cost of production of any non-monopolized commodity. The figures of the average price of copper and the disbursements of copper companies show there is little variation over the years in the ratio that the average selling price of the metal bears to the amount paid out per pound. There being a lack of any consistent price-cost relationship, the

delusion has long existed that oil is cheap, whatever the price. Oil finally has assumed the natural tendencies of other commodities and hereafter demand and supply may be expected to exert undisputed control over price. Many believe that the oil industry itself is uninformed as to the cost of producing oil except in the most general way. In the future it will be insufficient to anticipate the successful development of an oil property merely because a quantity of oil can be obtained therefrom, for the over-all cost factor must first be satisfactorily answered. It is hoped that more thought will be directed to the study of oil costs beyond the present limited horizon and that cost data will occupy a proper place in company reports.

Economics of the Crude Oil Potential in the United States. By JOSEPH E. POGUE. (*Trans., Petr. Dev. and Tech.*, 1931. 3500 *words.*)—The shut-in crude oil potential is the summation of the surplus initial production brought in during the past three years. The potential has the economic effect of an inventory, the effect being approximately one-half that of the potential converted into actual production. The price of crude oil correlates with above-ground inventories corrected by adding thereto the inventory equivalent of the potential. If all drilling ceased, the potential would be sufficient to maintain crude oil production in balance with demand for 28 months. If all restrictions were removed and the oil could be handled physically, the crude petroleum output of the United States in 1931 would be 1.5 billion barrels. In such event, the price of Mid-Continent crude oil would have an indicated decline to 10¢ per barrel. The crude oil potential is still increasing, thus exerting pressure on crude oil markets. If a higher crude oil price level is desired by the petroleum industry, some method of reducing the crude oil potential must be found. The methods of control thus far practiced in the petroleum industry are faulty to the extent that they may have permitted the potential to increase. Any methods for stabilizing the petroleum industry that do not involve a reduction in the economic effect of the combined inventory are not fundamental and cannot be counted on to succeed.

Mining Geology

Value of Ceramic Tests in Subsurface Correlation of Cretaceous Shales in Central Wyoming. By C. E. DOBBIN and E. A. SWEDENBORG. (*Tech. Pub. No.* 424. 6000 *words.*)—Experiments carried out by the writers indicate that when Cretaceous shales in central Wyoming are subjected to burning tests in a combination muffle-fusion, gas-fired furnace under carefully controlled conditions they develop certain physical characteristics that appear to be diagnostic of the part of the geologic column from which they came. The results obtained from these tests are useful in identifying the position in the geologic section represented by cuttings of Cretaceous shales penetrated after drilling through a cover of Tertiary or later formations when the cuttings yield neither lithologic nor paleontologic criteria that will permit correlations to be made.

Geophysical Prospecting

Measurements of Resistivity by the Central Electrode Method at the Abana Mine, Northwestern Quebec, Canada. By LACHLAN GILCHRIST. (*Tech. Pub. No.* 386. 4000 *words.*)—At the Abana, northwestern Quebec, there is a porphyritic quartz diabase dike about 225 ft. wide and extending in a north-south direction, of which the location has been determined in general outline by means of a dip needle or magnetometer. This dike has an overburden of glacial drift 75 to 200 ft. thick. Adjacent to the dike on the western side there is an orebody of pyrite and chalcopyrite and sphalerite of low resistivity and on the eastern side there is another orebody of considerably higher resistivity. A central current electrode was placed at points above the dike and end current electrodes were placed at points about 1000 ft. east and

1000 ft. west of the central electrode. Electrodes for potential measurements were placed 50 ft. apart along lines running north and south through the central current electrode stations. With this layout measurements of resistivity at successive distances from the central electrodes were made. From our interpretation of the results an approximate determination of the depth of the overburden and a fairly accurate determination of the location of the orebodies could be made.

Method for Determining the Magnetic Susceptibility of Core Samples. By WILLIAM M. BARRET. (*Tech. Pub. No.* 394. 12,000 *words.*)—The ballistic method described in this paper furnishes a convenient and accurate means for the rapid determination of the susceptibility of pulverized samples. While its order of sensitivity is below that of certain magnetic balance and magnetometric methods, nevertheless, it is entirely satisfactory for practical measurements related to magnetic surveys. Although modifications of the arrangement have been employed for years in commercial D. C. permeameters, the present design incorporates certain refinements which adapt it to the specific uses of the geomagnetic investigator. Unlike some of the classical methods the measurements are absolute rather than comparative and their accuracy is unaffected, to a sensible degree, by stray magnetic fields. A mathematical treatment of the theory and calibration for each method of operation is included. The discussion of the apparatus is clarified by photographs and performance curves. The present method was developed to meet the need for routine analysis of specimens in the geophysical laboratory where it was felt that general utility, ruggedness, ease of manipulation and reproducibility of results were of greater importance than extreme sensitivity.

Mathematical Theory of Electrical Flow in Stratified Media with Horizontal, Homogeneous and Isotropic Layers. By DAVID O. EHRENBURG and ROBERT J. WATSON. (*Tech. Pub. No.* 400. 8500 *words.*)—Making use of a generalized method of images the authors have obtained a solution for the electrical flow in a stratified media with horizontal, homogeneous, and isotropic layers. This solution can be used for any number of layers, of any thickness, and of any resistivity. Theoretically, the generalized method of images is applicable to any number of layers. However, in practice it will be found that in the case of a more than three-layer earth this method becomes awkward. The method here presented is analytical, although certain principles of the image theory are made use of indirectly in the building of the tentative solution. The differential equations finally lead to a set of algebraic equations; these result in formulas which can be arranged in a rather simple form, so that particular solutions for arbitrary cases may be built up and carried out numerically by one unfamiliar with both the methods of images and the calculus. Six calculated curves are included which show the percentage anomaly of resistivity for the cases of a two-layer, three-layer, and four-layer earth, using fixed current electrodes and moving potential electrodes.

Electrical Exploration Applied to Geological Problems in Civil Engineering. By E. G. LEONARDON. (*Tech. Pub. No.* 407. 6500 *words.*)—The author first discusses the technique of measuring the resistivity of the ground by potential methods. He indicates that two different procedures can be utilized according to the geological problem at hand, namely, the horizontal exploration and the electrical vertical drilling. In the horizontal exploration a uniform depth of investigation is employed and enables the establishment of resistivity profiles or resistivity maps. In the electrical drilling measurements are made at different depths of investigation at a given station. This makes it possible to study the constitution of the underground along the vertical of the station. The second part of the article deals with practical examples of field exploration by both techniques. At Bridge River, B. C., a few resistivity profiles enabled the prediction of the position of a zone of soft rock with water occurrences, along the tunnel line. At Morrisburg, Ontario, electrical vertical drillings were used

to prepare a topographical map of the bedrock at a proposed dam site on the St. Lawrence River. At Masson (Que.) the same procedure was employed to ascertain whether or not there was a safe cover of solid rock over a tunnel which was contemplated at a dam site on the Lièvre River.

A Magnetic Method of Estimating the Height of Some Buried Magnetic Bodies. By A. S. EVE. (*Tech. Pub. No.* 408. 2500 *words.*)—Magnetic methods are known for roughly estimating the depth underground of the top of a thin, long, magnetic dike, but it is more difficult to estimate the depth below ground of the bottom. By taking measurements on the ground, and on a raised wooden platform, the resulting difference of the magnetic intensity was measured with an Askania vertical magnetometer, at the Falconbridge nickel mine near Sudbury. The mathematical theory is given, and some pioneer measurements are discussed. The theory for broad and for inclined dikes, due to L. V. King, is also outlined.

A New Development in Electrical Prospecting. By Hans Lundberg and Theodor Zuschlag. (*Tech. Pub. No.* 415. 7000 *words.*)—A new development in the procedure of electrical prospecting, permitting potential drop ratio determinations, is described and its fundamental applications discussed. The paper concludes with a discussion of its application at Buchans, Newfoundland, Askesta, Sweden, and Range, Alberta.

A Theoretical Study of Apparent Resistivity. By J. N. HUMMEL. (*Tech. Pub. No.* 418. 15,000 *words.*)—A number of surface potential methods and the relation of potential distribution to subsurface resistivity are discussed. The principal field of application of these methods is the investigation of strata with horizontal (or nearly horizontal) bedding planes. Absolute potential differences are measured along a line at the surface. Of all possible arrangements of electrodes five are listed which are most suitable from the standpoint of field procedure and clearness of interpretation. From the potential differences, the subsoil resistivity, termed apparent resistivity, may be computed. It changes with the distance between the secondary electrodes and is also a function of the resistivities of the surface formations as well. The cases of a spherical mass, a two-layer problem, and a three-layer problem, are analyzed. In the layer problems, the author makes use of the theory of images. From the potential, the expression for the apparent resistivity is derived as a function of the distance between the secondary electrodes. Several curves are given for different ratios of resistivity. In the three-layer problem, an approximate curve is computed which approaches the true curve for large distances and the true curve can be found by graphical interpolation. This procedure greatly simplifies the mathematical analysis.

Least Squares in Practical Geophysics. By IRWIN ROMAN. (*Preprint.* 23,000 *words.*)—The method of least squares has been successfully used in practical geophysics to improve the interpretation of the observed data by reducing the effects of inaccuracies in the measurements. This paper discusses problems selected from: (1) sound surveying, (2) seismograph reflection exploration, (3) torsion balance surveying. For each case, the general problem is formulated and the results are applied to numercal data to illustrate the actual forms of calculation.

A. I. M. E. Technical Publications, 1931

Separates of all the *Technical Publications* published in 1931 are available at Institute headquarters. All the papers are on file in public, university and technical libraries, and when so indicated in the list, may be found in the TRANSACTIONS.

Copies of these papers have been mailed to those members of the Institute who are registered in the different classes: A-Metal Mining; B-Milling and Concentration; C-Iron and Steel Division; D-Nonferrous Metallurgy; E-Institute of Metals Division; F-Coal Division; G-Petroleum Division; H-Nonmetallic Minerals; I-Mining Geology; K-Mining Administration; L-Geophysical Prospecting. Other members may obtain copies by writing to the Secretary. Additional copies are sold at the rate of 1 cent per page; minimum charge, 25 cents.

METAL MINING

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE*
411-A.43	Ventilation at the Portovelo Mines, Ecuador (16)	J. P. Harmon	New York, Feb., 1931	**T31**, 83
425-A.44	Use of Models for the Study of Mining Problems (28)	P. B. Bucky	New York, Feb., 1931	

MILLING AND CONCENTRATION

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE*
375-B.32	Ball Mill Studies, I (53)	A. W. Fahrenwald H. E. Lee	New York, Feb., 1931	
401-B.33	Experimental Flotation of Oxidized Silver Ores (9)	H. S. Gieser	New York, Feb., 1931	
410-B.34	Flotation of Minor Gold in Large-scale Copper Concentrators (32)	E. S. Leaver J. A. Woolf	New York, Feb., 1931	
414-B.35	Effect of Particle Size on Flotation (23)	A. M. Gaudin J. O. Groh H. B. Henderson	New York, Feb., 1931	
416-B.36	Ball Mill Studies, II.—Thermal Determinations of Ball Mill Efficiency (13)	A. W. Fahrenwald G. W. Hammar H. E. Lee W. W. Staley	New York, Feb., 1931	

IRON AND STEEL DIVISION

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE*
380-C.59	Open-hearth Steel Process as a Problem in Chemical Kinetics (34)	E. R. Jette	New York, Feb., 1931	**C31**, 80, *111*
381-C.60	Thermodynamic Study of the Phasial Equilibria in the System Iron-carbon (26)	Yap, Chu-Phay	New York, Feb., 1931	
382-C.61	Influence of Dissolved Carbide on the Equilibria of the System Iron-carbon (19)	Yap, Chu-Phay	New York, Feb., 1931	
383-C.62	Research on Blast-furnace Slags. Three papers (68)	R. S. McCaffery Co-workers	New York, Feb., 1931	
384-C.63	Statistical Analysis of Blast-furnace Data (13)	R. S. McCaffery R. G. Stephenson	New York, Feb., 1931	**C31**, 60, *70*

* Volume numbers are in boldface; page numbers of papers, Roman; and page numbers of discussions, italics. Where no number is given, paper has not appeared in TRANSACTIONS. Discussions are printed following the respective papers.

The significance of the volume numbers is given on page 621.

The number in parentheses following each title indicates the number of pages in the *Technical Publication.*

Iron and Steel Division.—*Continued*

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
385-C.64	Air Discharge of Circular Tuyeres (10)	R. S. McCaffery D. E. Krause	New York, Feb., 1931	**C31**, 72
388-C.65	Bright Annealing of Steels in Hydrogen (9)	F. C. Kelley	New York, Feb., 1931	**C31**, 378, *384*
402-C.66	Inclusions and Their Effect on Impact Strength of Steel (33)	A. B. Kinzel W. Crafts	New York, Feb., 1931	**C31**, 140, *173*
404-C.67	Some Notes on Blue Brittleness (11)	L. R. van Wert	New York, Feb., 1931	**C31**, 230, *238*
409-C.68	Carbon-oxygen Equilibrium in Liquid Iron (13)	H. C. Vacher E. H. Hamilton	New York, Feb., 1931	**C31**, 123, *134*
412-C.69	Austenite-pearlite Transformation and the Transition Constituents (6)	A. Sauveur	New York, Feb., 1931	**C31**, 247, *250*
419-C.70	Age-hardening of Austenite (25)	F. R. Hensel	Boston, Sept., 1931	**C31**, 255, *278*
421-C.71	On the Art of Metallography (36)	F. F. Lucas	New York, Feb., 1931	**C31**, 11
426-C.72	Low-carbon Steel (31)	H. B. Pulsifer	Boston, Sept., 1931	**C31**, 347, *375*
427-C.73	Magnetic Properties Versus Allotropic Transformations of Iron Alloys (7)	T. D. Yensen N. A. Ziegler	Boston, Sept., 1931	**C31**, 313, *317*
428-C.74	Composition Limit of the Alpha-gamma Loop in the Iron-tungsten System (7)	W. P. Sykes	Boston, Sept., 1931	**C31**, 307, *311*
430-C.75	Some Developments in High-temperature Alloys in the Nickel-cobalt-iron System (21)	C. R. Austin G. P. Halliwell	Boston, Sept., 1931	
434-C.76	Dilatometric Study of Chromium-nickel-iron Alloys (23)	V. N. Krivobok M. Gensamer	Boston, Sept., 1931	**C31**, 325, *345*
435-C.77	Transformational Characteristics of Iron-manganese Alloys (19)	H. Scott	Boston, Sept., 1931	**C31**, 284, *300*
436-C.78	Inclusions and Their Effect on Impact Strength of Steel, II (10)	A. B. Kinzel W. Crafts	Boston, Sept., 1931	**C31**, 181, *188*
439-C.79	Development of Continuous Gas Carburizing (17)	R. J. Cowan	Boston, Sept., 1931	**C31**, 386, *400*
440-C.80	Method for Electrolytic Extraction of MnO, MnS, FeS and SiO_2 Inclusions from Plain Carbon Steels (15)	G. R. Fitterer	Boston, Sept., 1931	**C31**, 196, *208*

Nonferrous Metallurgy

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
378-D.29	High-silica Retorts at the Rose Lake Smelter (6)	G. L. Spencer, Jr.	New York, Feb., 1931	**T31**, 119, *122*
379-D.30	Extraction of Tantalum and Columbium from Their Ores (18)	C. G. Fink L. G. Jenness	New York, Feb., 1931	**T31**, 147
389-D.31	Failures of Cast-iron Kettles in Lead Refining (13)	C. E. Swartz	New York, Feb., 1931	**T31**, 125, *135*
403-D.32	Arsenic Elimination in Reverberatory Refining of Native Copper (16)	C. T. Eddy	New York, Feb., 1931	**T31**, 104, *117*

Institute of Metals Division

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page
390-E.125	Plasticity of Copper-zinc Alloys at Elevated Temperatures (9)	A. Morris	New York, Feb., 1931	**E31**, 336, *340*
391-E.126	Die Pressing of Brass and Copper Alloys (8)	J. R. Freeman, Jr.	New York, Feb., 1931	**E31**, 329, *335*
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